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# INTERNATIONAL POPLAR RIVER WATER QUALITY STUDY

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Appendix E: Plant, Mine and Reservoir Operations

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## Appendix E: Plant, Mine and Reservoir Operations

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Prepared by the Plant, Mine and Reservoir  
Operations Committee of the International  
Poplar River Water Quality Board,  
International Joint Commission

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May, 1979

Mr. E. H. G. Cornford  
Co-Secretary, Canada  
International Poplar River Water Quality Board

Dear Mr. Cornford:

We are pleased to transmit herewith the final report of the Plant, Mine, and Reservoir Operations Committee entitled "Poplar River Power Project, Project Features, Water Pollution Sources, and Water Pollution Control." This report contains the final revisions of text and incorporates the most recent available data on studies undertaken for the International Poplar River Water Quality Board by its committee on Plant, Mine, and Reservoir Operations.

This report includes a description of the Poplar River Power Plant, the Coronach Mine, and the Cookson Reservoir as well as ancillary structures and features. It details anticipated effects of the project on the Poplar River, and presents recommendations to mitigate adverse project effects. Annexes to this report contain the engineering and analytical data used in the determination of probable consequences of project operation.

D. Wayne Draper  
D. Wayne Draper, Canadian Co-Chairman  
Plant, Mine, and Reservoir Operations  
Committee

Lee R. Rice  
Lee R. Rice, U. S. Co-Chairman  
Plant, Mine, and Reservoir Operations  
Committee

SUMMARY OF CONTENTS

MAIN REPORT	PROJECT FEATURES, WATER POLLUTION SOURCES AND WATER POLLUTION CONTROL
ANNEX A	DETAILED MINING PLAN
ANNEX B	PREPARATION, DISTRIBUTION AND ANALYSIS OF POPLAR RIVER LIGNITE AND ASH SAMPLES
ANNEX C	WATER ANALYSIS - PUMPWELL WATER QUALITY DATA
ANNEX D	MINE SPOIL AND RESERVOIR SOIL LEACHING TEST RESULTS
ANNEX E	POPLAR RIVER ASH LEACHING TEST RESULTS
ANNEX F	BOUNDARY DAM ASH LAGOON STUDY RESULTS
ANNEX G	ASH LAGOON SEEPAGE STUDY
ANNEX H	PLANT AND RESERVOIR TECHNICAL DATA
ANNEX I	EVAPORATION FROM COOKSON RESERVOIR
ANNEX J	RESERVOIR OPERATION MODEL DOCUMENTATION AND SELECTED MODEL OUTPUT LISTINGS

## PREFACE

This report, prepared by the Plant, Mine and Reservoir Operations Committee, is the result of studies and analyses undertaken between January, 1978 and May, 1979. Individual contributors have worked during this period to gather, evaluate, revise and present in report form relevant information on the Poplar River power project.

The purpose of the report is to describe the features of the planned Poplar River power development, to determine what effects operation of the development will have on the East Poplar River, and to suggest measures which will mitigate potential adverse effects identified.

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TABLE OF CONTENTS - MAIN REPORT

	Page
TITLE PAGE	( i )
SUMMARY OF CONTENTS	( ii )
PREFACE	( iii )
LIST OF TABLES	( ix )
LIST OF FIGURES	( xi )
LIST OF PHOTOGRAPHS	( xi ii )
LIST OF ABBREVIATIONS AND CONVERSION FACTORS	( xiv )
1. SUMMARY	1-1
2. INTRODUCTION	2-1
2.1 Project Location and Status	2-1
2.2 Objectives of Study	2-1
2.3 Organization of Report	2-4
2.4 Acknowledgements	2-4
3. MINING	3-1
3.1 Description	3-1
3.1.1 Introduction	3-1
3.1.2 Characteristics of Topsoil and Overburden	3-6
3.1.3 Characteristics of Hart Seam Lignite	3-8
3.1.4 Characteristics of Underlying Formations	3-9
3.2 Mining Plan	3-12
3.2.1 Mining Rate-Determining Factors	3-12
3.2.2 Short-Term Mining Plan	3-13
3.2.3 Long-Term Mining Plan	3-13
3.3 Reclamation Plan	3-17
3.4 Mining Equipment and Facilities	3-17
3.4.1 Mining Equipment	3-17
3.4.2 Loadout Facilities	3-17
3.4.3 Maintenance Shop and Fueling area	3-17

TABLE OF CONTENTS - MAIN REPORT (Continued)

	Page
3.5 Mining Procedures	3-20
3.5.1 Drilling and Sampling	3-20
3.5.2 Dewatering	3-20
3.5.3 Topsoil Removal and Storage	3-21
3.5.4 Excavation of Overburden and Spoil Placement	3-22
3.5.5 Lignite Mining Operations	3-22
3.5.6 Reclamation	3-26
3.6 Mining Inputs and Losses to Surface Water Systems	3-28
3.6.1 Dewatering	3-28
3.6.2 Mine Pit and Sump Drainage	3-39
3.6.3 Diversion around Mine Site	3-32
3.6.4 Non-Point Source Discharge from Reclaimed Areas	3-34
3.6.5 Sanitary and Other Wastes	3-37
3.7 Mining Inputs and Losses to Groundwater Systems	3-39
3.7.1 Dewatering	3-39
3.7.2 Mine Pit and Sump Drainage	3-39
3.7.3 Leachates from Spoil Materials and from Reclaimed Land	3-39
3.7.4 Sanitary and Other Wastes	3-41
3.8 Deposition from Mining Emissions to the Atmosphere	3-41
4. TRANSPORTATION SYSTEMS	4-1
4.1 Description	4-1
4.2 Inputs and Losses to Surface Water Systems	4-1
4.3 Inputs and Losses to Groundwater Systems	4-2
4.4 Transportation System Emissions	4-2
5. POWER PLANT	5-1
5.1 Description	5-1
5.1.1 General	5-1
5.1.2 Condenser Cooling Water System	5-1
5.1.3 Auxiliary Cooling Water System	5-16
5.1.4 Water Treatment Plant	5-17
5.1.5 Oil Tank Storage and Oil Transport Lines	5-18
5.1.6 Transformers	5-18
5.1.7 Boiler	5-19

TABLE OF CONTENTS - MAIN REPORT (Continued)

	Page
5.1.8 Powerhouse Drains and Sumps	5-22
5.1.9 Outside Storm Drains and Roof Drains	5-24
5.1.10 Sanitary and Solid Wastes	5-24
5.1.11 Holding Pond	5-26
5.1.12 Coal Storage, Handling and Characteristics	5-28
5.1.13 Ash Disposal System	5-34
 5.2 Inputs and Losses to Surface Water Systems	 5-47
5.2.1 Station Water Balance	5-47
5.2.2 Condenser Cooling Water Discharge	5-52
5.2.3 Auxiliary Cooling Water Discharge	5-54
5.2.4 Plant Roof and Yard Drains	5-55
5.2.5 Reverse Osmosis Rejects	5-55
5.2.6 Ash Lagoon Discharge	5-56
 5.3 Inputs and Losses to Groundwater System	 5-69
5.3.1 Sanitary Waste Disposal	5-69
5.3.2 Coal Storage Area	5-69
5.3.3 Seepage from Ash Lagoons	5-70
5.3.4 Seepage from Holding Pond	5-73
 5.4 Deposition from Atmospheric Emissions	 5-75
5.4.1 General	5-75
5.4.2 Deposition from Stack Emissions	5-75
5.4.3 Deposition from Fugitive Dust Emissions	5-78
5.4.4 Effect on Surface Waters	5-80
 6. RESERVOIR OPERATIONS	 6-1
6.1 Description	6-1
6.1.1 Location and Physical Characteristics	6-1
6.1.2 Method of Operation	6-1
6.1.3 Circulation and Mixing	6-8
6.1.4 Thermal Regime	6-10
6.1.5 Softening of Reservoir Water	6-11
6.1.6 Recreational Uses	6-14
6.2 Inputs and Losses to Surface Water Systems	6-15
6.2.1 General	6-15
6.2.2 Inputs to Cookson Reservoir	6-16
6.2.3 Losses from Cookson Reservoir	6-22

TABLE OF CONTENTS - MAIN REPORT (Continued)

	Page
7. VILLAGE OF CORONACH	7-1
7.1 Population	7-1
7.2 Water Supply	7-1
7.3 Wastewater Treatment	7-1
7.4 Solid Wastes	7-3
7.5 Drainage	7-4
7.6 Emissions	7-4
8. MITIGATION MEASURES	8-1
8.1 Suggested Quality Criteria	8-1
8.2 Evaluation of Proposed Operation	8-2
8.2.1 Description of Reservoir Operation Model	8-2
8.2.2 Components of International Boundary Flow	8-4
8.2.3 Modelling Assumptions for Testing Sensitivity	8-5
8.2.4 Relative Significance of Contributing Effluent Sources	8-7
8.2.5 Results of Model Runs	8-7
8.2.6 Cost of Proposed System	8-12
8.3 Identified Mitigation Alternatives	8-14
8.3.1 Mining Operations	8-14
8.3.2 Transportation Systems	8-14
8.3.3 Ash Disposal System	8-14
8.3.4 Other Power Plant Wastewaters	8-16
8.3.5 Power Plant Cooling System	8-16
8.3.6 Reservoir Operation	8-16
8.3.7 Village of Coronach	8-17
8.3.8 Other Alternatives	8-17
8.4 Evaluation of Selected Mitigation Alternatives	8-17
8.4.1 Mining Operation Alternatives	8-17
8.4.2 Ash Disposal System Alternatives	8-20
8.4.3 Control of Seepage from Ash Lagoons	8-25
8.4.4 Effectiveness of Ash System Mitigation Alternatives	8-27
8.4.5 Supplemental Mitigation Options for TDS Reduction	8-30
8.4.6 Comparison of Minimum Reservoir Levels with Selected Mitigation Alternatives	8-38
8.4.7 Mitigation to Meet Other Concerns	8-41
8.5 Recommended Mitigation Measures	8-43
9. REFERENCES	9-1

LIST OF TABLES

<u>Table No.</u>	<u>Title</u>	<u>Page</u>
1-1	Estimated Annual Cost of Mitigation (2 Units)	1-3
3-1	Physical Characteristics of Coronach Mine Lignite	3-8
3-2	Hart Seam Lignite, Major Components and Analyses Summary	3-10
3-3	Mining Progress Estimates, Block 1, 1979-1983	3-15
3-4	Lignite Reserve Estimate, Mining Block 1	3-16
3-5	Lignite Reserve Estimate, Coronach Mine	3-16
3-6	Hart Seam Dewatering Well Water Quality Data	3-29
3-7	Monthly Seepage Volume into Mine Pit	3-30
3-8	Test Pit #2 Water Quality Data	3-31
3-9	Runoff Rate Estimates for the Coronach Mine Area	3-33
3-10	Statistical Summary of Precipitation Records for Swift Current, 46-Year Period from 1931-76	3-33
3-11	Readily Soluble Salts in Overburden	3-39
3-12	Chemical Analyses of Leachate Solutions	3-40
3-13	Particulate Emissions from Mine Related Activities	3-42
3-14	Emissions from Mine Vehicular Traffic	3-43
3-15	Particulate Deposition Rates Associated with Mining Activity	3-43
4-1	Coal Train Emissions	4-2
5-1	Elemental Analysis - Coronach Area Lignite	5-33
5-2	Summary - Coal Consumption and Ash Production	5-36
5-3	Constituent Concentrations - Poplar River Ash	5-37
5-4	Ash Lagoon Statistics	5-40

LIST OF TABLES (Continued)

<u>Table No.</u>	<u>Title</u>	<u>Page</u>
5-5	Poplar River G.S. Water Balance - 1 Unit	5-50
5-6	Poplar River G.S. Water Balance - 2 Units	5-51
5-7	Results of Static Leaching Tests on Poplar River Ash	5-58
5-8	Results of Coal Leachate Tests	5-62
5-9	Inputs to Ash Lagoons and Typical Lagoon Water Quality During an Extended Dry Period	5-64
5-10	Boundary Dam Ash Lagoon Study Results	5-66
5-11	Column Leachate Test Results - Poplar River Ash	5-74
6-1	Temperature Distribution in Cookson Reservoir	6-13
6-2	Present Use (1975) Flow in East Poplar River at International Boundary	6-17
6-3	Readily Soluble Boron and Total Salts in Reservoir Area Soil Samples	6-21
6-4	Monthly Natural Gross Evaporation (mm) from Cookson Reservoir	6-25
6-5	Monthly Precipitation (mm) in East Poplar River Basin	6-28
6-6	Forced Evaporation Loss from Cookson Reservoir	6-30
8-1	Comparison of Loadings from Different Effluent Sources	8-8
8-2	Results of Reservoir Operation Model Runs for Proposed System	8-9
8-3	Results of Reservoir Operation Model Runs for Primary Mitigation Options	8-28
8-4	Results of Reservoir Operation Model Runs for Supplemental Mitigation Options	8-33

LIST OF FIGURES

<u>Figure No.</u>	<u>Title</u>	<u>Page</u>
2-1	Site Location Map, Poplar River Power Project	2-2
2-2	Major Components of the Poplar River Power Project	2-3
3-1	35-Year Mining Plan, Coronach Mine, 1979-2014	3-2
3-2	Mine Service Area and Lignite Loadout Facility	3-3
3-3	Geologic Section across Coronach Mine	3-7
3-4	Thickness of Hart Seam Lignite at the Coronach Mine Site	3-11
3-5	Mining Block 1, Short-Term Mining Plan, 1979-1983	3-14
3-6	Schematic Diagram of Mining Procedure at Coronach Mine	3-23
3-7	Schematic Diagram of Selective Spoil Placement	3-24
3-8	Schematic Diagram of Spoil Reclamation at the Coronach Mine	3-27
3-9	Runoff from Mine Area	3-35
3-10	Estimated Sediment Yield from Coronach Mine	3-36
5-1	Location of Major Power Plant Features	5-3
5-2	Layout of Poplar River Generating Station	5-5
5-3	Location of Cooling Water Intake and Oil Storage Area	5-7
5-4	Plan and Profile of Cooling Water Intake	5-8
5-5	Plan and Profile of Cooling Water Pumphouse	5-9
5-6	Cooling Water Discharge Outlet Structures	5-14
5-7	Plan View - Powerhouse Basement Level	5-20
5-8	Profile of Powerhouse Unit #2	5-21
5-9	Holding Pond - Plan and Profile	5-27
5-10	Ash Lagoon Profiles	5-41

LIST OF FIGURES (Continued)

<u>Figure No.</u>	<u>Title</u>	<u>Page</u>
5-11	Section F-F of Ash Lagoon Dyke and C.W. Discharge Channel	5-42
5-12	Subsurface Profile along Probable Seepage Path from Ash Lagoons (Section E-E, Figure 5-1)	5-43
5-13	Poplar River G.S. Water Balance - 1 Unit	5-48
5-14	Poplar River G.S. Water Balance - 2 Units	5-49
5-15	Isopleths of SO <sub>2</sub> Mean Annual Concentration for 600 MW	5-77
5-16	Isopleths of Mean Annual Particulate Concentrations for 600 MW	5-79
6-1	Morrison Dam and Outlet Works	6-2
6-2	Area and Capacity Data for Cookson Reservoir	6-4
6-3	Discharge Rating Curves - Morrison Dam Spillway and Riparian Outlet	6-6
6-4	Sections Across Cookson Reservoir	6-9
6-5	Reservoir Zones for Computing Thermal Regime	6-12
6-6	Baseflow Determination for East Poplar River at International Boundary	6-18
6-7	Gross Evaporation Estimates for Cookson Reservoir 1931-76	6-26
8-1	Plot of Monthly Reservoir Levels - 48 Years	8-13
8-2	10-Year Running May to September Mean TDS Levels at International Boundary for Selected Computer Model Runs	8-39
8-3	10-Year Running May to September Mean Boron Levels at International Boundary for Selected Computer Model Runs	8-40

LIST OF PHOTOGRAPHS

<u>Photo No.</u>	<u>Title</u>	<u>Page</u>
3-1	Power Shovel Extracting Lignite near Boundary Dam Power Station	3-4
3-2	Small Dragline Removing Overburden at Mine near Boundary Dam Power Station	3-5
3-3	Bucket for Dragline at the Saskatchewan Power Corporation Coronach Mine	3-18
3-4	Walking Dragline under Construction at the Saskatchewan Power Corporation Coronach Mine	3-19
5-1	Poplar River G.S. - Partially Constructed Power-house in May, 1978	5-2
5-2	Outlet of Cooling Water Discharge Pipe to Cooling Water Discharge Channel During Construction in May, 1978	5-15
5-3	View of Empty Sewage Lagoon during Power Plant Construction in May, 1978	5-26
5-4	Holding Pond Filled with Local Runoff during Plant Construction in May, 1978	5-29
5-5	Evaporation Pond during Power Plant Construction in May, 1978	5-31
5-6	Coal Handling Silo during Construction in May, 1978	5-32
5-7	View looking Southward from the Station of the Area to be Occupied by the Ash Lagoons - May, 1978	5-39
6-1	Morrison Dam Showing Service Spillway and Riparian Outlet	6-3

LIST OF ABBREVIATIONS AND CONVERSION FACTORS  
 (Common symbols for elements and chemical compounds have not been listed)

Abbreviations

ac	acre	mg	milligram
ac-ft	acre-feet	ml	millilitre
AES	Atmospheric Environment Service	mm	millimetre
BOD	Biological Oxygen Demand	mps	metre per second
BTU	British Thermal Unit	MW	megawatt
cfs	cubic feet per second	PFRA	Prairie Farmer Rehabilitation Administration
cm	centimeter	pH	Negative log of hydrogen ion concentration
DOE	Department of the Environment	ppm	parts per million
EC	specific conductance	RG	Rockglen
EPA	Environmental Protection Agency	RO	reverse osmosis
FSL	full supply level	R.O.D.	release-on-demand
ft	feet	s	second
g	gram	SAR	sodium adsorption ratio
gal	gallon	SPC	Saskatchewan Power Corporation
G.S.	generating station	SRC	Saskatchewan Research Council
ha	hectare	T. Alk.	total alkalinity
hm	hectometre	TDS	total dissolved solids
hr	hour	TH	test hole
Igpd	Imperial gallon per day	ton	short ton
Igpm	Imperial gallon per minute	tonne	metric ton
IJC	International Joint Commission	TSP	total suspended particulates
in	inch	TSS	total suspended solids
IPRWQB	International Poplar River Water Quality Board	ug	microgram
kg	kilogram	U.S.	United States
kJ	kilojoule	U.S.gpd	U.S. gallon per day
km	kilometer	U.S.gpm	U.S. gallon per minute
kwh	kilowatt hour	yd	yard
l	litre	yr	year
lb	pound	°C	degrees Celsius
m	metre	°F	degrees Fahrenheit
MBAS	Methylene blue activated substance	°K	degrees Kelvin
MECO	Montreal Engineering Co. Ltd.		

Conversion Factors

ac-ft	= 1,233.5 m <sup>3</sup> = 1.2335 dam <sup>3</sup>
ac	= 4047 m <sup>3</sup> = 0.4047 ha
BTU/lb	= 2.325 kJ/kg
dam <sup>3</sup>	= 1000 m <sup>3</sup>
ft <sup>3</sup>	= 28.3171 l
ha	= 10,000 m <sup>2</sup> = 2.471 ac
hm	= 100 m = 328.08 ft
hm <sup>3</sup>	= 1 x 10 <sup>6</sup> m <sup>3</sup>
Igpm	= 0.0631 l/s
in	= 2.54 cm
kg	= 2.20462 lb
kJ	= 0.9471 BTU
km	= 0.62137 miles
l	= 0.03532 ft <sup>3</sup> = 0.21997 I. gal. = 0.26420 U.S. gal.
l/s	= 13.193 Igpm = 15.848 U.S. gpm
m	= 3.2808 ft
m <sup>2</sup>	= 10.7638 ft <sup>2</sup>
m <sup>3</sup>	= 1000 l = 35.3144 ft <sup>3</sup>
year	= 31,557,600 sec
tonne	= 1000 kg = 1.1023 ton (short)
U.S.gpm	= 0.0758 l/s

This report, prepared by the Plant, Mine and Reservoir Operations Committee, contains a full description of the Poplar River power development with particular emphasis on the quantity and quality of effluents expected to result from the coal mine and power plant operations. The potential deterioration in water quality crossing the International Boundary attributed to the various liquid effluent loadings, as well as stack and fugitive dust emissions and plant, mine and reservoir operational practices is assessed. Alternative water pollution control designs and practices are presented and discussed and recommendations are made on control or mitigation measures that will minimize deterioration in the quality of transboundary flow in the East Poplar River.

The information, evaluations and conclusions in this report result from extensive review of available information on the lignite coal mining and power plant operation as proposed by the Saskatchewan Power Corporation as well as numerous individual studies initiated and carried out by the Plant, Mine and Reservoir Operations Committee. At the commencement of the study, information was lacking in many areas including method of mine reclamation, leachability of constituents from the coal and ash, subsurface conditions in the proposed ash disposal area and details of the power plant waste management system. The new studies initiated filled in many of the information gaps but information remains scarce in some areas. Certain design and operational aspects have been in a state of flux during the course of the study and although the descriptions of proposed systems are based on the best information available to the committee at the time the report was written, subsequent changes may have since altered some features.

The new studies initiated by the Plant, Mine and Reservoir Operations Committee included:

- (i) soil sample collection around the perimeter of Cookson Reservoir and a drilling program in the proposed ash disposal area.
- (ii) laboratory analysis of Poplar River coal and ash samples.
- (iii) laboratory leaching experiments on soil, mine spoil, coal ash and coal samples.
- (iv) independent assessment of potential seepage losses from the ash lagoons.
- (v) a comparative ash lagoon study at the Boundary Dam generating station including sampling and laboratory analysis.

- (vi) independent assessment of evaporation losses from Cookson Reservoir.
- (vii) modification of the reservoir operation computer model used by the Saskatchewan Power Corporation.
- (viii) numerous reservoir model computer runs to test the sensitivity of various mitigation alternatives and assist in the determination of mitigation requirements.

All of the above studies or investigations were totally or partially conducted either by outside contract arranged by the committee or by the Saskatchewan Power Corporation. These studies were supplemented by numerous other studies and assessments by individual committee members including development of a detailed mining plan, quantification and characterization of effluents and extensive direct use of the reservoir operation model to assess mitigation alternatives.

As a result of its studies and investigations, the committee has identified potential problem areas and developed recommendations designed to prevent unacceptable deterioration in the water quality of transboundary flow. The major contributing source to surface water quality deterioration with the Poplar River development is the proposed once-through ash lagoon system. The element of primary concern with the ash lagoons is boron which was found to readily dissolve from the ash, but a fairly high dissolution rate for sulphate could also contribute to total dissolved solids build up in the reservoir. Seepage from the ash lagoons to the East Poplar River below Morrison Dam could be significant without some corrective action due to the inconsistency and fractured nature of the upper till layer in the ash disposal area.

Evaporation loss from Cookson Reservoir is a major factor in total dissolved solids build up in the reservoir, with natural evaporation being a greater contributor than forced evaporation from power plant operation. With the adoption of mitigation measures to control the contributions from the ash lagoons, the mine dewatering water becomes the largest source of total dissolved solids. Also, new consideration must be given to a power plant waste management system designed to operate in conjunction with the selected ash disposal alternative.

As a result of its evaluation of potential mitigation options to resolve the identified water pollution problem areas, the Plant, Mine and Reservoir Operations Committee concludes that feasible and practical mitigation measures are available that will permit recommended water quality objectives for the International Boundary to be met. The estimated annual costs of specific mitigation alternatives, and the total dissolved solids and boron levels that can be achieved at the border with those alternatives, are listed in Table 1-1.

The committee feels that a dry flyash/dewatered bottom ash system offers the greatest assurance that detrimental effects from ash disposal will be prevented. However, flexibility in selection of mitigation alternatives should not be totally precluded by specifying specific pollution control design or operational procedures in all cases. The following mitigation measures will ensure acceptable quality of trans-boundary flow in the East Poplar River:

TABLE 1-1 ESTIMATED ANNUAL COST OF MITIGATION (2 UNITS)

MEAN/MAXIMUM MAY TO SEPTEMBER CONCENTRATION AT INTERNATIONAL BOUNDARY (46 YEAR PERIOD)		MITIGATION MEASURES		ESTIMATED ADDITIONAL ANNUAL COST <sup>1</sup> PERCENT OF TOTAL PROJECT <sup>2</sup>
TDS	BORON			DOLLARS
<u>Proposed System</u>				
953/1795	8.1/18.7	Proposed ash disposal system (once-through lagoon)	0	0.00
<u>Primary Mitigation (Ash Disposal)</u>				
953/1795	1.8/2.5	Proposed ash disposal system with boron removal from decant	623,960 - 1,595,550	0.94 - 2.41
961/1754	2.9/5.6	Combined ash recirculating lagoon	87,720 - 645,880	0.13 - 0.97
963/1655	1.7/3.1	Dry flyash/dewatered bottom ash	1,128,060	1.70
<u>Supplemental Mitigation (Ash Disposal plus Additional TDS Reduction)</u>				
853/1375	2.9/5.4	Combined ash recirculating lagoon, lime softening of Cookson Reservoir	405,100 - 963,260	0.61 - 1.45
825/1218	2.1/3.1	Combined ash recirculating lagoon, water storage on Poplar River	194,670 - 752,830	0.29 - 1.14
733/1300	2.2/4.1	Combined ash recirculating lagoon, water storage above Cookson Reservoir	154,560 - 712,720	0.23 - 1.07
825/1218	1.7/3.1	Dry flyash/dewatered bottom ash, water storage on Poplar River	1,235,010	1.86
733/1300	1.7/3.1	Dry flyash/dewatered bottom ash, water storage above Cookson Reservoir	1,194,900	1.80

1 The two cost figures listed are for the proposed ash lagoon lining and an improved lining.  
If under drainage is provided, the improved lining costs may be higher than listed.

2 Base annual cost of total project (2 units) used in computations is \$66,300,000.

- (i) the ash disposal system should be designed and treatment should be provided so that any decant or blowdown releases are at reservoir water quality or better and total seepage from the ash disposal area to the East Poplar River and Cookson Reservoir do not exceed 2.0 l/s and 5.0 l/s respectively. The achievability of these seepage rates should be demonstrated and a monitoring program for ash disposal area seepage should be established.
- (ii) when the reservoir water quality first exceeds 1000 mg/l after spring freshet, remedial action in the form of a lime softening facility or facilities to separately contain and divert Poplar River or East Poplar River spring flow to meet the East Poplar River demand release should be provided.
- (iii) the power plant waste/management system should be redesigned in conjunction with the selected ash disposed option to provide containment and treatment to prescribed effluent limits for all wastewaters. Specific features requiring attention are lining of wastewater retention ponds, improved containment for transformer oil and seepage prevention for the sewage irrigation area.
- (iv) efforts should be made to minimize mine dewatering quantities, and practices and procedures designed to minimize the effects of other mine effluents should be adopted.
- (v) facilities should be provided to ensure that Fife Lake overflows can be prevented.
- (vi) power plant planned outages should be during the higher evaporation loss months whenever possible.
- (vii) a bilateral inspection group should be established with participation of water pollution control agencies from the two countries and the Saskatchewan Power Corporation to examine the final waste management system design plans and facilities.

Further details regarding these recommendations are contained in Section 8.5.

## 2. INTRODUCTION

### 2.1 Project Location and Status

The Poplar River Power Development is located in south central Saskatchewan, immediately north of the Canada-United States International Boundary as shown on Figure 2-1. The major features of the development are a lignite open pit mine, a thermal power plant, and a cooling reservoir formed by construction of Morrison Dam on the East Poplar River. Associated with these major features are the lignite storage and transport facilities from the mine to the power plant and an increased permanent and temporary population in the area, particularly in the village of Coronach. Each of these development features is outlined in Figure 2-2.

The current scheduling for the Poplar River project calls for commissioning of the first power unit on March 1, 1980. This means that some pre-commissioning operation would occur during the winter of 1979-80, requiring that the ash disposal system for the plant be constructed and operational by the fall of 1979. Construction of the first unit powerhouse is now partially completed and will be finalized during the summer of 1979. The second unit is expected to come on line in 1982, and has been identified by the Poplar River - Nipawin Board of Inquiry as one of Saskatchewan's most desirable future power development options.

The initial cut in the coal mining area has been started and lignite removal will commence during the summer of 1979.

Construction of Morrison Dam was completed in 1976 and Cookson reservoir was filled to its full supply level (FSL) of 753.0 m by containment of runoff during 1977, 1978 and 1979. Although water quality in the reservoir was poor during the first stages of reservoir filling (partially because of overflow from Fife Lake), good quality spring freshet water in 1978 and 1979 has resulted in relatively good reservoir water quality at present.

### 2.2 Objectives of Study

The Plant, Mine, and Reservoir Operations Committee was established under the International Poplar River Water Quality Board to identify, examine and assess those design and operational aspects of the Saskatchewan Power Corporation's Poplar River Power Development that might contribute to deterioration in the quality of surface or groundwaters in the United States portion of the Poplar River basin. Specifically, the committee's instructions from the Board were as follows:

"Identify, quantify, and chemically characterize all wastewater and runoff streams associated with the plant, mine and reservoir operation for the sequential time phases of construction, development and operation. The adequacy of wastewater treatment and disposal and runoff containment shall also be examined."

The effluent and other pollutant source information was to be provided for a "600 megawatt thermal power station."

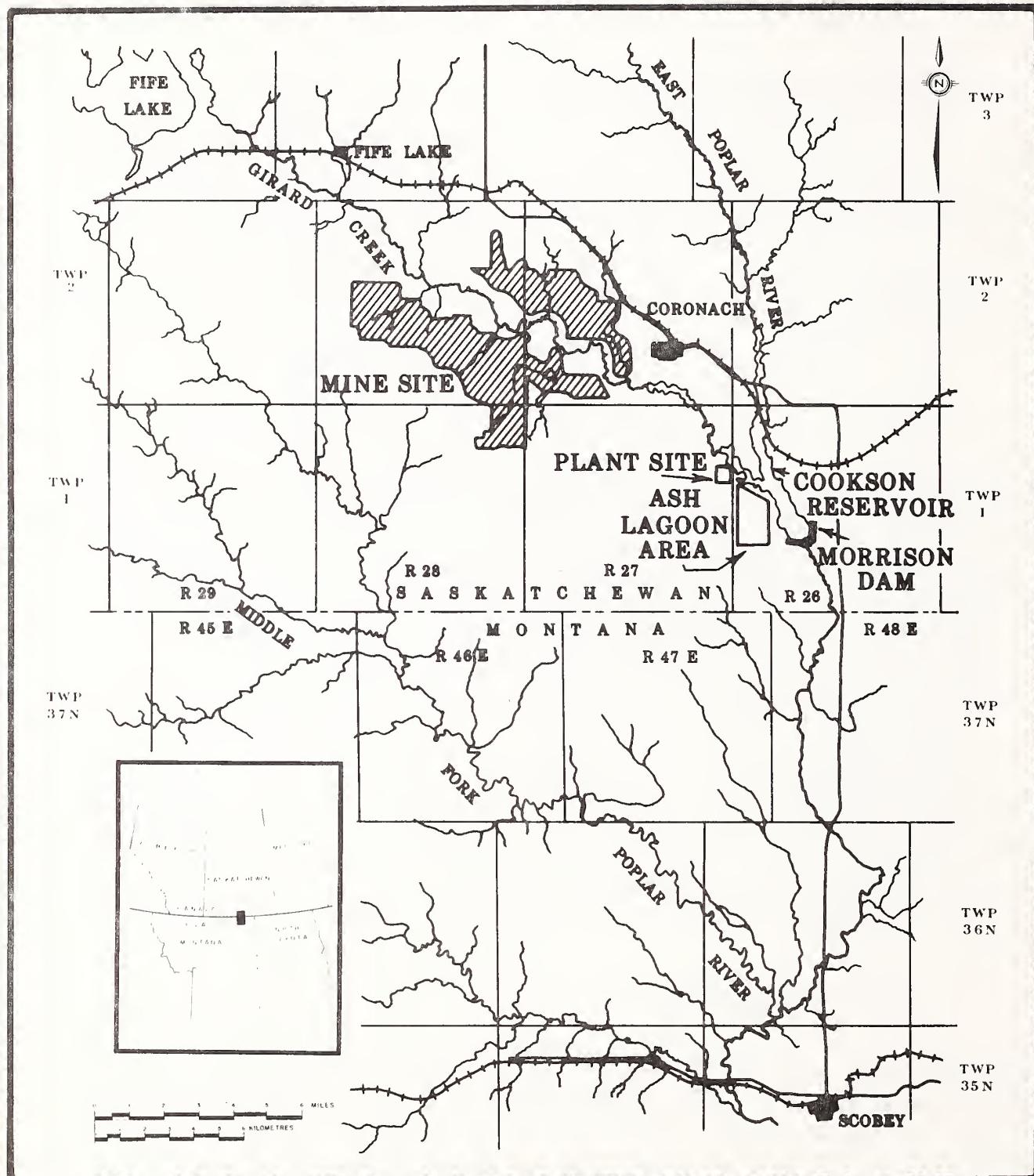


Figure 2-1. Site location map, Poplar River Power Project.

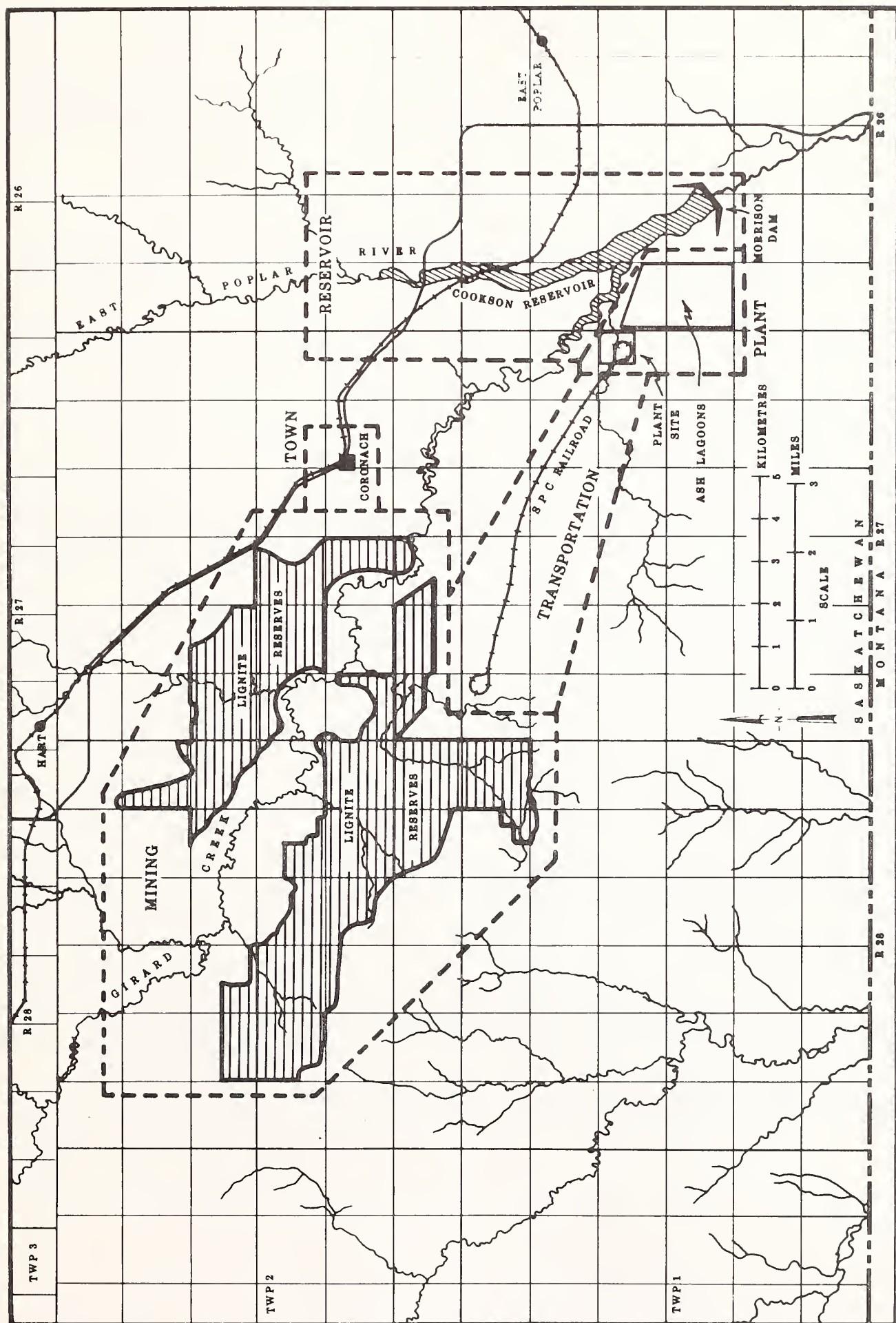


Figure 2-2. Major components of the Poplar River Power Project.

## 2.3 Organization of Report

This report is organized to deal with each of the major project features sequentially (Sections 3 through 7) with a follow-up concluding section on mitigation (Section 8). Each of Sections 3 through 7 is divided into two parts:

- (i) description of the project feature and operational aspects
- (ii) estimate of quantity and quality of inputs and losses to surface and groundwater systems.

If details of the project design and operation are not required, the reader may wish to refer only to the second parts of Sections 3 through 7 describing inputs and losses, and to Section 8 on mitigation of the major impacts identified.

## 2.4 Acknowledgements

The committee is indebted to their employing agencies for permitting them to participate in this important work and for the support staffs and agency resources dedicated to the study. We especially wish to acknowledge the cooperation and support of the Saskatchewan Power Corporation and their engineering consultants for supplying project data and for performing independent checks on the work of committee members.

In addition, the committee wishes to acknowledge the extensive analytical work undertaken on its behalf by the U.S. Environmental Protection Agency's Environmental Research Laboratories in Athens, Georgia and Corvallis, Oregon, and by the University of Wisconsin Water Chemistry Laboratory. Thanks is extended to Mr. T.L. Thoem, U.S. Environmental Protection Agency for air emission modelling information provided and to Mr. J.C. Brydges, Environmental Protection Service, Environment Canada for his work with the reservoir operation computer model. Special acknowledgement also is due Professor Gordon Pagenkopf of the Chemistry Department, Montana State University. Leaching studies and leachate analyses contracted to Dr. Pagenkopf were completed in a timely and excellent manner.

Other contributors through contract to the Plant, Mine and Reservoir Operations Committee were Geraghty and Miller, Inc., James F. MacLaren Limited, J.E. Edinger Associates Inc., Montreal Engineering Company Limited, CDM/Accu-Labs, Barringer Research Limited, Technical Service Laboratories and Technitrol Canada Limited.

### 3.1 Description

#### 3.1.1 Introduction

Lignite mining for the Poplar River Power Project will be carried out on lands owned by the Saskatchewan Power Corporation (SPC) Northwest of the plant site to the north and south of Girard Creek as shown in the 35-year mining plan (see Figure 3-1). No structures or permanent roadways are incorporated in the mine pit; all office space, equipment, and fuel storage facilities will be situated near the lignite loadout area as shown in Figure 3-2.

Topsoil in the area to be mined will be removed to a depth of about 15 cm (6 in), and will be stockpiled for later use in reclamation. After the topsoil has been removed, a dragline will be used to excavate overburden and cast it outside of the mining area. Overburden removal will continue until the lignite seam is exposed in the bottom of the cut for a width of 37 m (120 ft). The dragline then will move parallel to the axis of the cut and will continue removing overburden and exposing additional lignite.

Power shovels will be used to remove the lignite from the exposed seam and to load it into dump trucks that will haul the material to the loadout facility for processing and shipment by train to the power plant. Photo 3-1 shows a power shovel operating in the SPC mine near Estevan, Saskatchewan.

Mining proceeds in the long cut until the property boundary is reached, until the lignite thins so that recovery is no longer practical, or until the lignite seam is buried too deeply to permit economic recovery of the resource. The dragline then is moved so that a second cut, parallel to the first, can be made in the opposite direction. Overburden from the second cut is cast into the bottom of the first cut from which the lignite has been removed. In this manner, lignite is produced at a fairly uniform rate and mining proceeds in a series of long straight or gently curving cuts, leaving behind a series of long ridges of piled spoil ready for the commencement of reclamation activities. A relatively small dragline with a  $6.9 \text{ m}^3$  ( $9 \text{ yd}^3$ ) capacity is shown in Photo 3-2 removing overburden at the SPC mine near Estevan, Saskatchewan.

Reclamation of mined areas will consist of grading the spoil ridges, placement of suitable subsoils in some areas, contouring and final grading, spreading of stored topsoil, then seeding of the restored land surface.

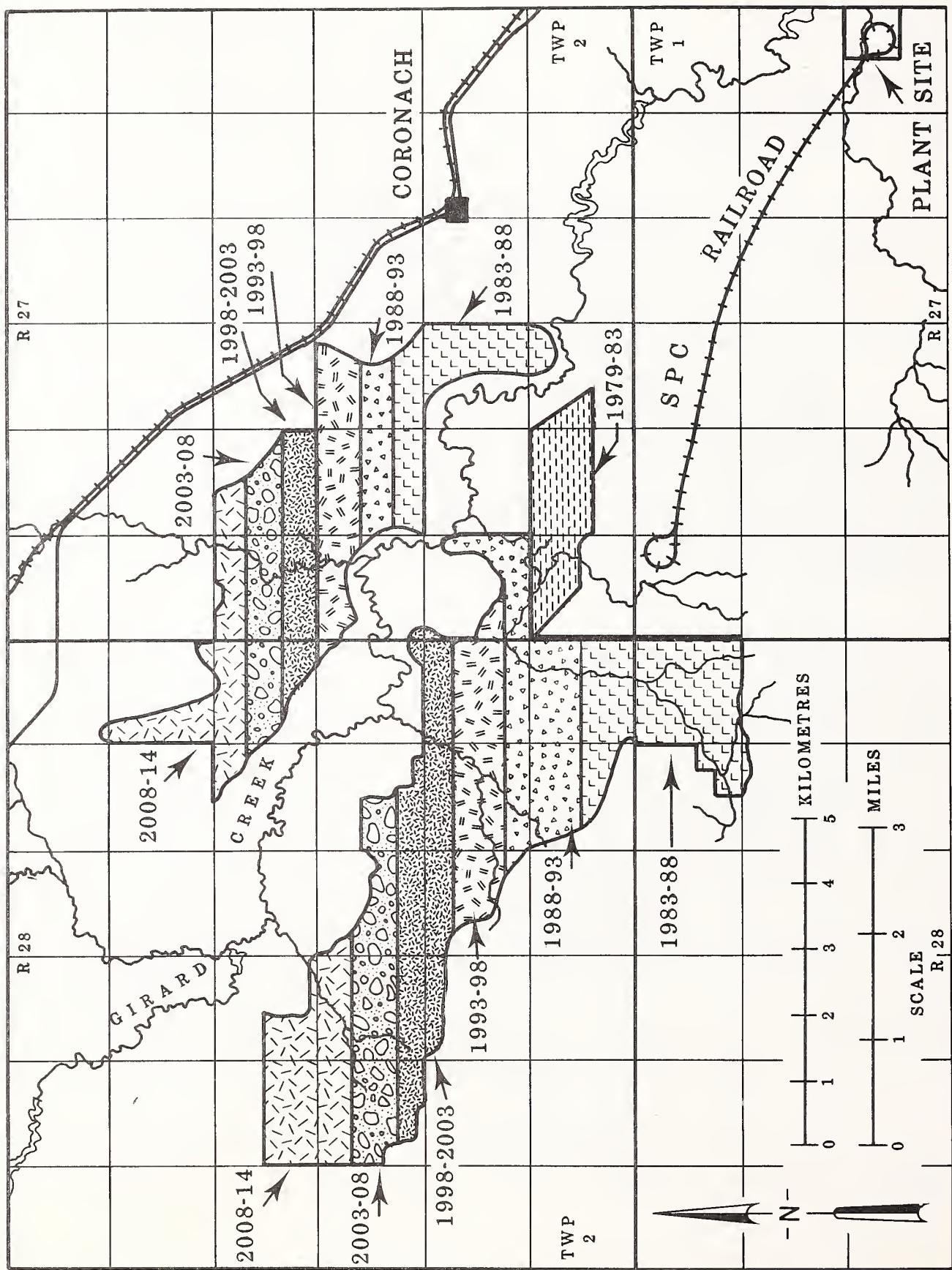


Figure 3-1. 35-Year mining plan, Coronach mine, 1979-2014.

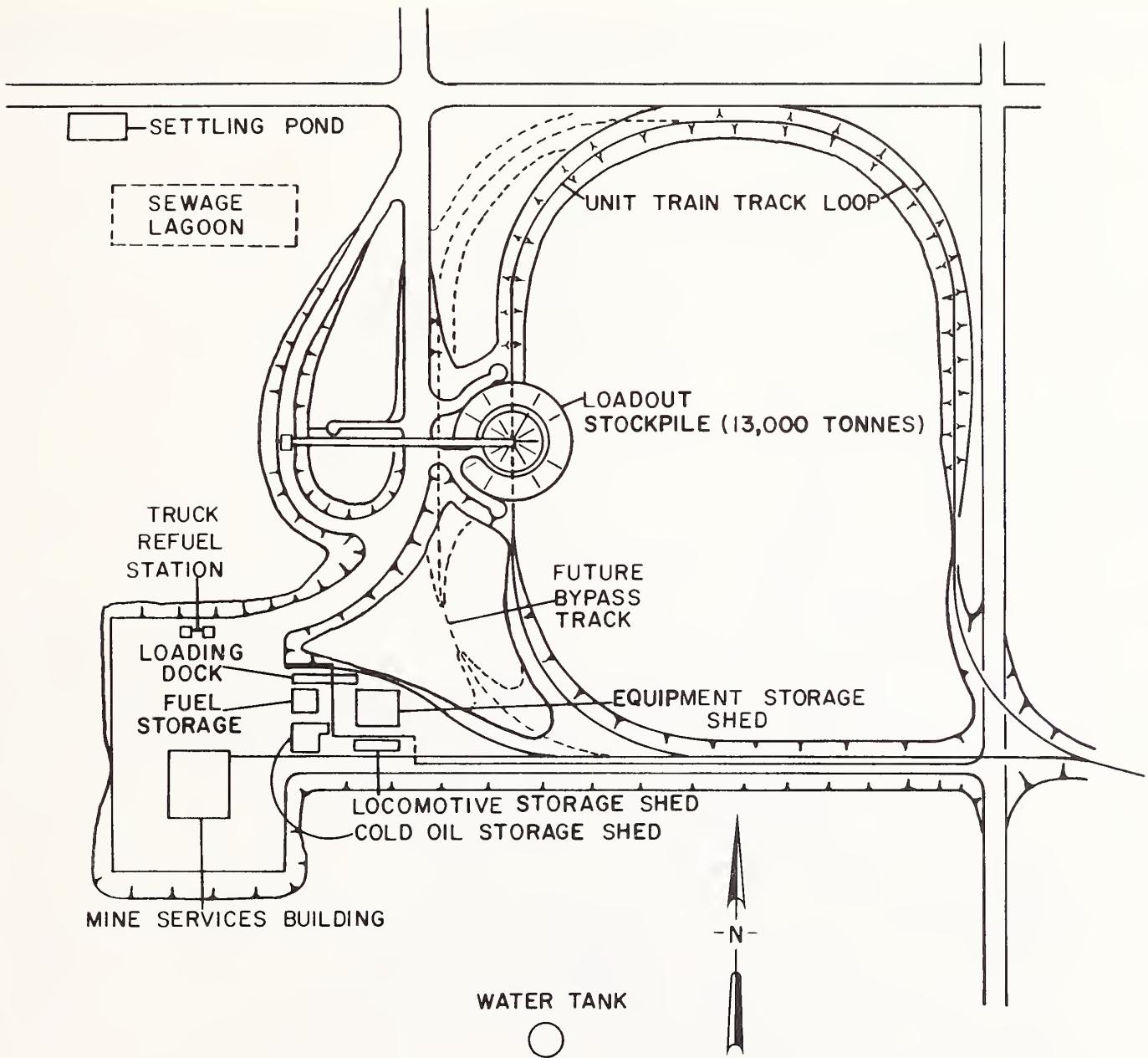


Figure 3-2. Mine service area and lignite loadout facility.

PHOTO 3-1 POWER SHOVEL EXTRACTING LIGNITE FROM THE ESTEVAN SEAM IN SPC MINE  
NEAR BOUNDARY DAM POWER STATION, ESTEVAN, SASKATCHEWAN



PHOTO 3-2 SMALL  $6.9 \text{ m}^3$  ( $9 \text{ yd}^3$ ) DRAGLINE REMOVING OVERBURDEN AT THE SASKATCHEWAN POWER CORPORATION MINE NEAR ESTEVAN, SASKATCHEWAN



### 3.1.2 Characteristics of Topsoil and Overburden

#### (a) Topsoil Characteristics

Soils at the Coronach mine site consist mainly of Chernozemic Brown soils with light brown to grayish-brown surface horizons and a relatively low organic matter content (Nelson et al, August, 1977). Some minor Gleysolic soils also occur there, but are confined to shallow kettle holes and slough localities.

These soils were derived from glacial till and from underlying bedrock; they are relatively thin and are moderately calcareous. The combined thickness of "A" and "B" horizons ranges from less than 0.3 m (1 ft) to more than 0.6 m (2 ft). Calculations involving topsoil removal and storage assume a 0.3 m (1 ft) depth and a bulking factor (swell factor) of 25 percent.

#### (b) Overburden Characteristics

##### (i) Composition and Geology

Till, sandy and clayey silt, very fine sand, clay and small amounts of gravel comprise the overburden overlying the lignite of the Hart Seam in the Coronach mine area (Ground Engineering, June, 1976, Part I). This area was covered by continental ice sheets during the Pleistocene epoch, and ice movement caused thrusting and mixing of the near-surface materials. In addition, Tertiary age Upper Ravenscrag Formation comprises the bedrock beneath the glacial drift. This formation is made up of thin and discontinuous interfingering lenticular beds of lignite, clays, sands, and silts. Thus, the overburden at the Coronach mine site is highly variable in composition. Figure 3-3 shows a representative geologic section in the Coronach mine area.

##### (ii) Overburden Thickness

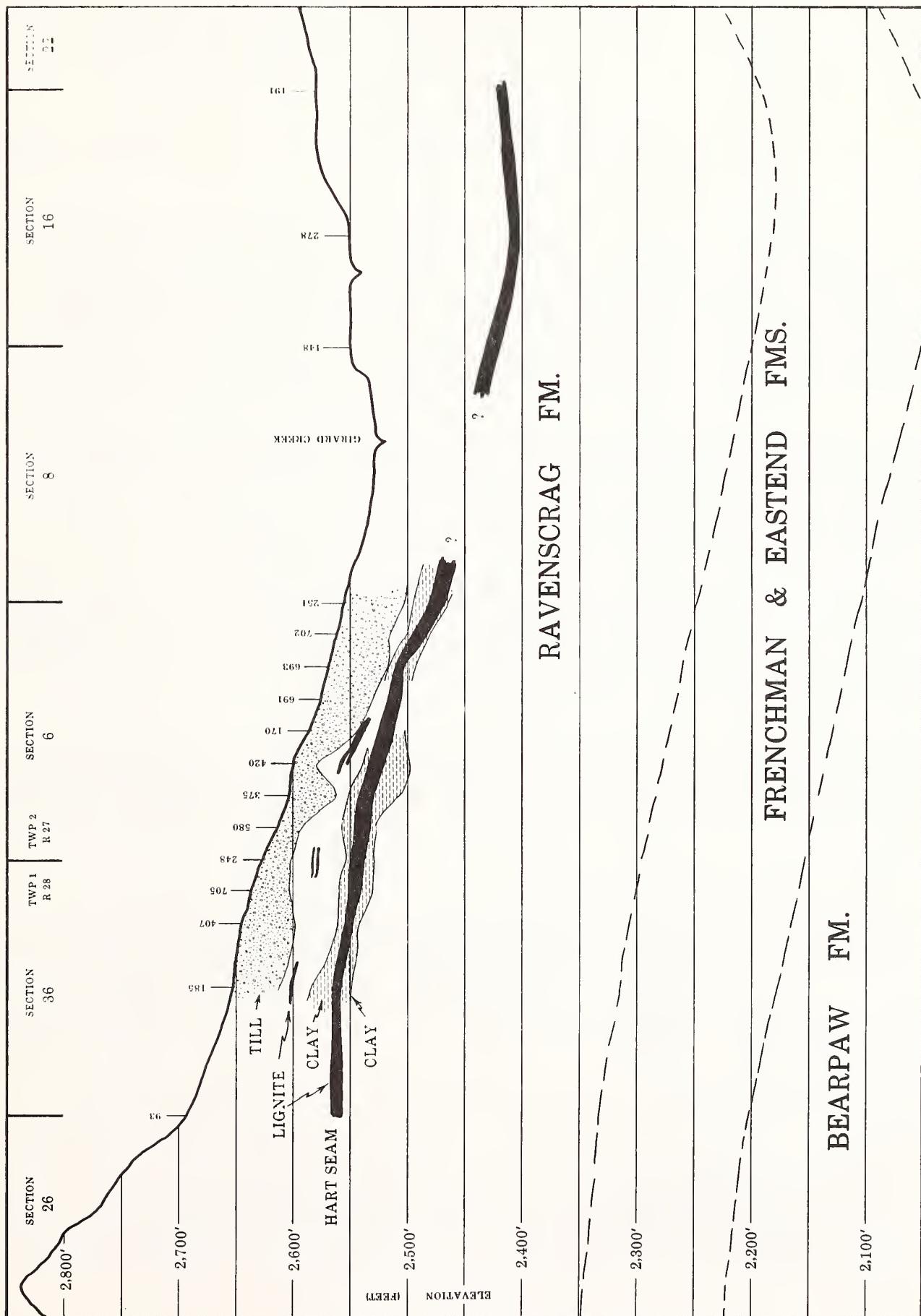
The overburden varies in thickness as well as in composition in the Coronach mine area. It is thinnest beneath Girard Creek and thickens toward the northeast and the southwest, reaching more than 61 m (200 ft) in thickness near Hart and also to the southwest of the mine site. The maximum overburden that will be excavated at the Coronach mine is 46 m (150 ft).

##### (iii) Engineering Properties

The Saskatchewan Power Corporation has excavated three test pits in the mine area to test slope stability in pit walls and in spoil material, and to determine the swell factor of excavated material (Ground Engineering, June, 1976, Part II-A). In addition, these pits were used to determine ground-water conditions, pit bottom conditions, and excavatability of the overburden material.

Undisturbed overburden was found to stand at grades up to 100 percent without excessive sloughing or slumping. Cuts in overburden have therefore been designed with 45-degree inclined walls.

Figure 3-3 Geologic Section across Coronach Mine



The angle of repose of spoiled overburden from the test pits was found to range from 25 degrees to 57 degrees, and to average about 34 degrees. This average value has been used in calculations to determine the size and shape of mine spoil piles at the Coronach mine site.

### 3.1.3 Characteristics of Hart Seam Lignite

#### (a) Physical Properties

Lignite from the Hart Seam near Coronach is typical of the Great Plains lignites; it is a low-sulfur, high ash, very high moisture, moderate heat content fuel. The lignite occurs in the Tertiary age Ravenscrag Formation and is immediately overlain and underlain by clay. The Hart Seam is an aquifer and the lignite is very moist when first exposed. The lignite checks and slacks badly when exposed to the air for long periods of time, and uncompacted piles of the material may ignite spontaneously.

When first mined, lignite from the Coronach mine is observed to have a moisture content of about 62.0 percent\* and exhibits a density of approximately 1,297 kg/m<sup>3</sup> (81.0 lb/ft<sup>3</sup>). In addition, it was observed from the test pit program at the Coronach mine site that the material had a bulking factor (swell factor) of 21.8 percent.

Tests on the 454-tonne (500 short tons) lignite sample consumed at the Boundary Dam Power Station showed an average moisture content of only 36.55 percent (SPC, May 5, 1976); thus, a considerable loss in weight and moisture content of the lignite can be expected to occur in routine operation of the Coronach mine loadout facility.

Using the information derived from the test pit program and from the test burn, it is possible to calculate some of the physical characteristics of the lignite in the seam, after mining, and when ready for use as a boiler fuel. These characteristics are summarized in Table 3-1.

Table 3-1 Physical Characteristics of Coronach mine lignite

	<u>Volume (m<sup>3</sup>)</u>	<u>Moisture Content (percent)</u>	<u>Density (kg/m<sup>3</sup>) (lb/ft<sup>3</sup>)</u>	
Lignite in place	1.000	62.0	1,581	98.7
Mined lignite	1.219	62.0	1,297	81.0
Processed lignite	1.219	36.55	777	48.5

\*This result is from a very limited sampling. Saskatchewan Power Corporation personnel indicate that recent samples show values ranging from 40 to 50 percent moisture in freshly mined lignite at the Coronach Mine.

(b) Chemical Composition

A large number of chemical analyses of lignite from the Hart Seam near Coronach are available. These analyses are summarized in Table 3-2.

Additional work on trace-element analysis of the lignite and its ash has been undertaken in order to supplement and to verify analyses already performed for the Saskatchewan Power Corporation. These results are presented in Annex B and summarized in Section 5.1.12(c).

(c) Thickness and Distribution

Portions of the Hart Seam lignite were eroded away before the period of glaciation and deposition of glacial and glaciofluvial debris on the exposed erosion surface of the Ravenscrag Formation. Thus, in addition to the variability in character and thickness imposed by the environment of lignite deposition, additional variability in thickness has resulted from the pre- and post-glacial periods.

Lignite of the Hart Seam ranges up to 5.2 or 5.5 m (17 or 18 ft) in thickness, but usually is only 2.4 to 3.7 m (8 to 12 ft) thick in the Coronach mine area. Partings of a few centimeters to more than 60 cm also are present in some areas. Where thin, the partings may be ignored and the entire section of lignite can be mined; but where thicker, the partings represent a significant dilution of the lignite and necessitate selective mining. Figure 3-4 shows the thickness of Hart Seam lignite in the Coronach mine area as determined by exploratory drilling.

3.1.4 Characteristics of Underlying Formations

The generalized geologic section shown in Figure 3-3 also shows the clay and other strata underlying the Hart Seam in the Ravenscrag Formation. Although these strata will not be excavated in the mining operation, they will exert a powerful influence on the mining procedures at the Coronach mine.

When first exposed in an excavation, the underclay is a hard, dense, and impermeable clay-shale. If this material is kept dry, it forms a good surface on which to operate haulage trucks, loaders, dozers, and power shovels, but the clay becomes very plastic when moistened and may prove unable to support haulage traffic in the pit bottom when such conditions are allowed to exist (Ground Engineering, June, 1976, Part II-A).

The impermeability of this underclay may result in other mining difficulties because of water pressure in underlying formations. The pit bottom may swell or heave due to distress from haulage traffic and from the unloading of overburden removal. Any connection with underlying aquifers under artesian pressure also could result in excessive water in the pit, making lignite extraction difficult and increasing haulage and pumping costs.

Table 3-2. HART SEAM LIGNITE, Major Constituent Analysis Summary

REMARKS	MOISTURE %	PROXIMATE ANALYSIS*				ULTIMATE ANALYSIS*				HEATING* VALUE (BTU/lb)
		VOLATILE MATTER %	FIXED CARBON %	ASH %	CARBON %	NITROGEN %	HYDROGEN %	OXYGEN %	SULFUR %	
Hart Seam Lignite Analyses from the mine area; 62 analyses (Petrow, 1974).	43.8	35.5	21.9	50.9	0.39	4.1	22.5	0.81	8,600	
Test Burn Lignite from Test Pit #2; composite of 10 samples (CDM/Accu-Labs).	9.88		23.34							0.72 8,385
Test Burn Lignite; arithmetic mean of 7 individual analyses (U.S. Dept. of Energy Coal Lab.).	9.8	39.0	38.6	22.4	52.4	0.7	3.2	20.8	0.6	8,020
Test Burn Lignite; duplicate analyses of a single sample (CDM/Accu-Labs).	12.27		22.20							0.76 8,424
Test Burn Lignite; freshly mined from Test Pit #2 (Boundary Dam Power Station Lab.)	36.55†									
Willow Bunch Lignite Samples; arithmetic mean of 8 analyses (Montana Dept. of Health and Environmental Sciences)	8.0		16.0							0.9 9,138
Best Values (Average)	36.55†	43.7	35.6	21.4	50.9	0.40	4.1	22.5	0.80	8,596
Normalized Best Values	36.55	43.4	35.4	21.4	50.8	0.40	4.1	22.5	0.80	

\*All values except moisture content are reported on a moisture free basis.

†This value for the moisture content was selected as the most representative value because it represents a large sample of freshly mined material. All other moisture values shown in the table are from samples subjected to a varying period of air drying.

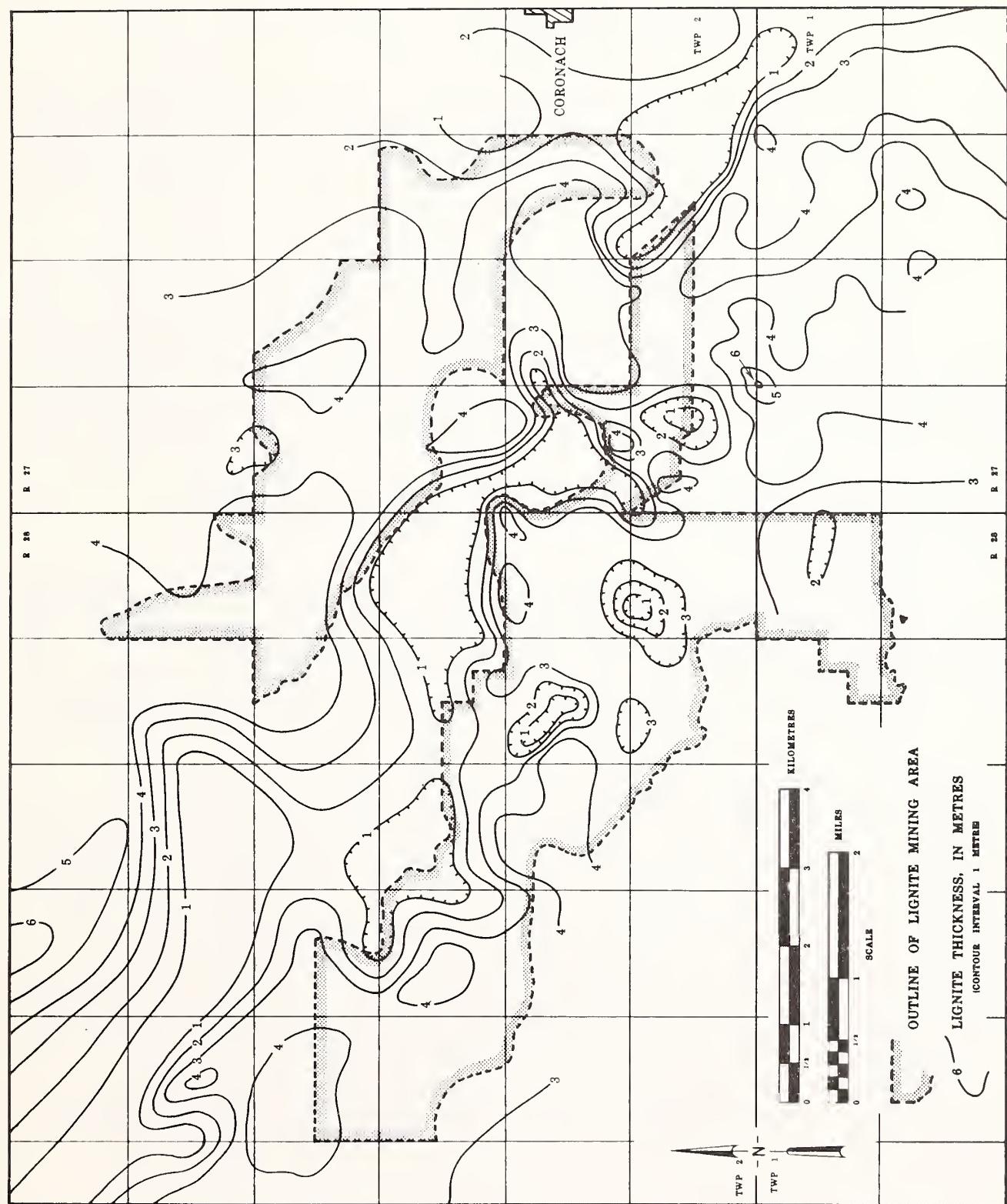


Figure 3-4. Thickness of Hart Seam lignite at the Coronach mine site.

### 3.2 Mining Plan

#### 3.2.1 Mining Rate-Determining Factors

The rate at which lignite mining is carried out is limited by the equipment used in the mining operation and by mine conditions such as depth of overburden, lignite thickness, water inflow rates, and ground stability. It will be necessary, however, to mine sufficient lignite at the Coronach mine to supply the needs of the Poplar River Power Station, and this demand provides the ultimate lignite mining rate-determining factor.

A 454-tonne lignite sample from the Hart Seam at the Coronach mine site was burned at the Boundary Dam Power Station on December 19, 1975, to determine the fuel characteristics of this material. The average moisture content of the lignite consumed at Boundary Dam was 36.55 percent, and the average heating value of this fuel was 12,567 kJ/kg (5,403 BTU/lb). However, the average heating value for Hart Seam lignite throughout the Coronach mine area is about 12,686 kJ/kg (5,454 BTU/lb) when calculated on an "as received" basis with an average moisture content of 36.55 percent. The calculated density of test burn lignite was 777 kg/m<sup>3</sup> (48.51 lb/ft<sup>3</sup>), yielding a calculated heating value of 9.858x10<sup>6</sup> kJ/m<sup>3</sup> (264,574 BTU/ft<sup>3</sup>).

Because of the swell factor experienced when mining this material (21.8 percent), the lignite in place in the seam will yield a value of 12.606x10<sup>6</sup> kJ/m<sup>3</sup> (338,329 BTU/ft<sup>3</sup>). Thus, one cubic meter of lignite in the Hart Seam will produce 12.606x10<sup>6</sup> kJ of heat energy when mined, processed, air dried, and burned in the Poplar River Power Station.

The required rate of heat input for the boilers at the Poplar River Power Station is specified to be 3.165x10<sup>9</sup> kJ/hr (3.00x10<sup>9</sup> BTU/hr) for each 300-MW unit. On a day-to-day basis, the two units at Coronach will be available approximately 87 percent of the time (except for a 30-day period each year when each unit will be shut down for maintenance and cleaning). Moreover, all of the lignite in the seam is not available for consumption at the power plant. A few centimeters of lignite at the top of the seam are lost to overburden removal and cleaning of the seam. Similarly, a few centimeters of lignite at the base of the seam usually are left to prevent mining into the underclay. In very flat seams of uniform thickness, the recovery of coal or lignite may reach 96 percent of the material present in the seam. The presence of partings in the seam, undulations in the surface of the seam, and variations in seam thickness result in lower recovery factors. Based on conditions in the Hart Seam at the Coronach mine, the estimated recovery factor will be approximately 85 percent.

With this information, the average daily lignite consumption for a single megawatt unit can be calculated in terms of unmined (in place) lignite volume.

$$\frac{3.165 \times 10^9 \text{ kJ/hr} \times 24 \text{ hr/day} \times 0.87}{(\text{heating rate}) \quad (\text{time}) \quad (\text{service factor})} = 6,168 \text{ m}^3/\text{day}$$

$$12.606 \times 10^6 \text{ kJ/m}^3 \times 0.85 \\ (\text{heating value}) \quad (\text{lignite recovery factor})$$

This corresponds to a volume of 217,800 ft<sup>3</sup>/day. Thus, at full operation with two units, the Poplar River Power Station will consume approximately 12,335 m<sup>3</sup> (435,600 ft<sup>3</sup>) of lignite reserves daily.

Mining rate calculations have been performed using volume units (m<sup>3</sup>) rather than weight units (tonnes) because of the large effect of lignite moisture content. For example, the calculated weight of processed lignite required to yield one million kilojoules is 78.81 kg. The weight of freshly mined lignite which will yield one million kilojoules after processing is 131.6 kg. The weight of lignite in the seam which will produce one million kilojoules after mining and processing (allowing for mining losses) is 184.4 kg. Thus, one tonne of lignite in place is equivalent to only about 427 kg of lignite at the power station, a weight loss of more than 57 percent.

### 3.2.2 Short-Term Mining Plan

Initially, lignite mining will be carried out in sections 4, 5, and 6 as shown in Figure 3-5. The progress of mining in this block has been calculated on the basis of fuel requirements for the Poplar River Power Station and the probable timetable of unit startup. Table 3-3 shows the projected lignite requirements for the power station, the progress of mining, and the cumulative disturbed area of the pit and spoil areas. Table 3-4 shows calculated lignite reserves of Block 1.

As can be seen from these tables, the lignite reserves of this block will be exhausted approximately one year sooner than originally planned. Thus, mining is expected to begin in Block 2S one year earlier than previously planned.

### 3.2.3 Long-Term Mining Plan

The economic life of the Poplar River Power Station is projected to be about 35 years. Lignite reserves necessary to fuel the plant throughout this period of time have been secured by the Saskatchewan Power Corporation. Figure 3-1 shows the ultimate extent of mining at the Coronach mine site. Table 3-5 lists the areas and lignite reserves of each mining block and the total area and lignite reserves of the mine as presently planned.

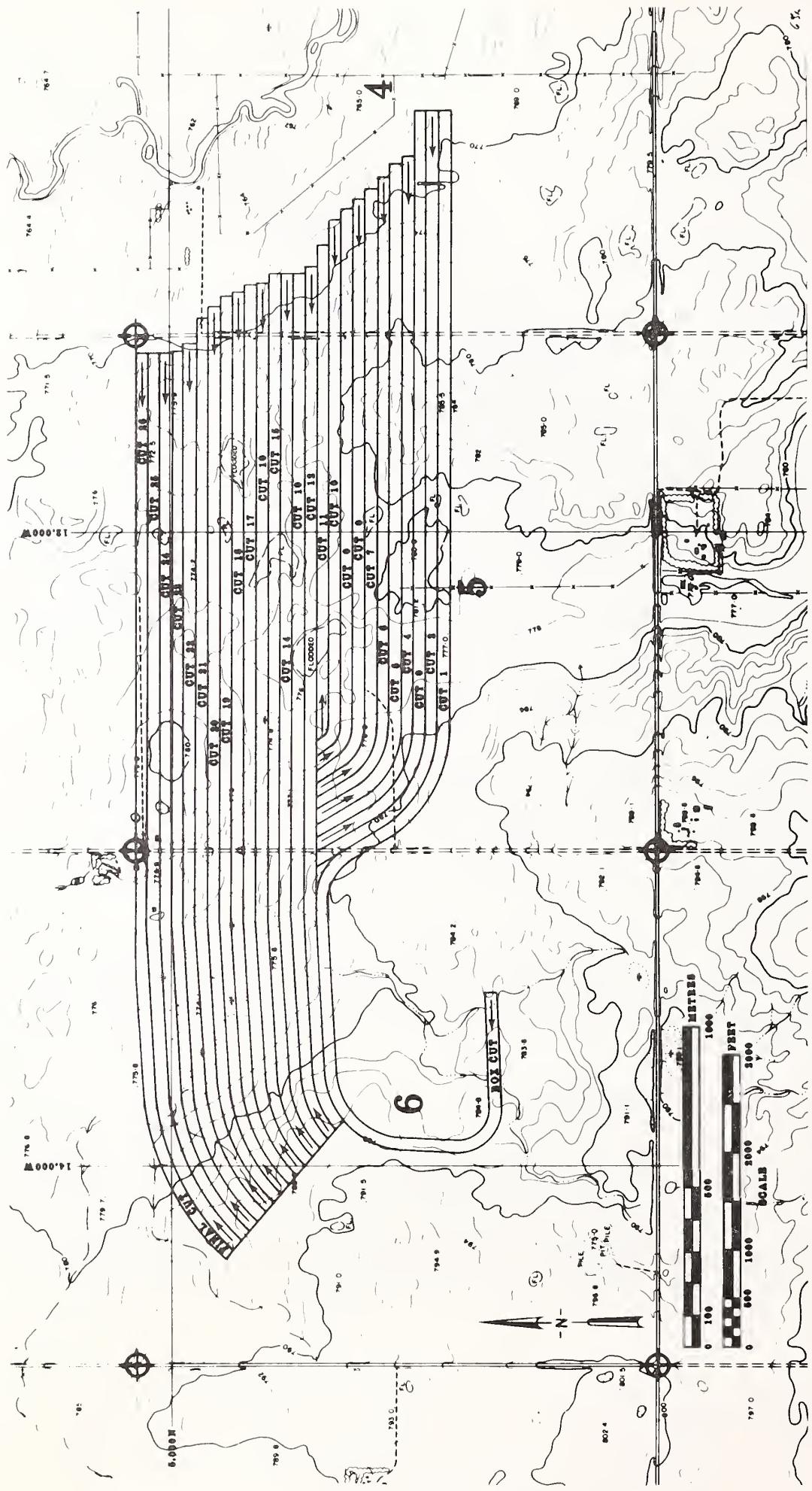


Figure 3-5. Mining block 1, short-term mining plan, 1979-1983.

TABLE 3-3. Mining progress Estimates, Block 1, 1979-1983.

YEAR	MONTH	LIGNITE VOLUME REQUIRED (m <sup>3</sup> )	CUT NO.	LENGTH MINED IN CUT (m)	TOTAL PIT AREA (hm <sup>2</sup> )	TOTAL SPOIL AREA (hm <sup>2</sup> )	TOTAL RECLAIMED AREA (hm <sup>2</sup> )	TOTAL DISTURBED AREA (hm <sup>2</sup> )
1979	April	51,650	1	440	3.88	6.35	0	10.23
	May	51,650	1	870	7.76	12.70	0	20.46
	June	51,650	1	1,310	11.64	19.05	0	30.69
	July	51,650	1	1,740	15.52	25.40	0	40.92
	August	51,650	1	2,180	19.40	31.75	0	51.15
	September	51,650	1	2,620	23.28	38.10	0	61.38
	October	51,650	1	3,050	27.16	44.45	0	71.61
	November	45,200	1	3,430	30.55	50.00	0	80.55
	December	46,700	1	3,820	34.06	55.74	0	89.80
	January	46,700	1	4,210	37.57	61.49	0	99.05
	February	43,690	2	320	38.33	62.17	0	100.50
	March	162,520	2	1,520	40.90	64.28	0	105.18
1980	April	157,280	3	200	42.72	66.62	0	109.34
	May	162,520	3	1,420	41.17	70.58	0	111.75
	June	0	3	1,420	41.17	70.58	0	111.75
	July	162,520	4	220	38.12	66.70	7.96	104.82
	August	162,520	4	1,460	38.25	63.38	15.92	101.93
	September	157,280	5	440	38.30	59.85	23.88	98.15
	October	162,520	5	1,690	38.21	56.35	31.84	94.56
	November	157,280	6	780	38.18	52.88	39.80	91.06
	December	162,520	7	10	38.18	57.64	39.80	95.82
	January	162,520	7	1,300	38.07	62.37	39.80	100.44
	February	146,800	8	520	37.98	66.69	39.80	104.67
	March	162,520	8	1,820	37.90	71.57	39.80	109.47
1981	April	157,280	9	1,240	37.94	69.78	46.32	107.72
	May	162,520	10	820	37.87	68.16	52.83	106.03
	June	0	10	820	37.87	61.64	59.35	99.51
	July	162,520	11	440	37.78	59.89	65.86	97.67
	August	162,520	12	130	37.74	57.92	72.38	95.66
	September	157,280	12	1,430	37.64	56.14	78.90	93.78
	October	162,520	13	80	36.93	52.78	85.41	89.71
	November	157,280	13	1,290	36.90	51.47	91.93	88.37
	December	162,520	14	10	36.86	56.85	91.93	93.71
	January	162,520	14	1,380	36.83	61.91	91.93	98.74
	February	146,800	14	2,620	36.80	66.47	91.93	103.27
	March	162,520	15	1,300	36.76	71.55	91.93	108.31
1982	April	157,280	15	2,630	36.73	66.72	101.68	103.45
	May	162,520	16	1,210	36.71	61.81	111.43	98.52
	June	45,200	16	1,570	36.70	53.41	121.18	90.11
	July	209,230	17	480	36.77	49.82	130.93	86.59
	August	209,230	17	2,120	37.09	46.11	140.68	83.20
	September	202,480	18	900	37.31	41.92	150.43	79.23
	October	325,050	19	570	37.55	40.76	160.18	78.31
	November	314,560	20	40	37.77	39.13	169.93	76.90
	December	325,050	20	2,400	37.95	47.53	169.93	85.48
	January	325,050	21	1,890	38.19	55.82	169.93	94.01
	February	293,590	22	1,140	38.36	63.28	169.93	101.64
	March	325,050	23	640	38.55	71.58	169.93	110.13
1983	April	314,560	24	110	38.79	71.58	178.05	110.37
	May	325,050	24	2,640	39.24	71.59	186.40	110.83
	June	157,280	25	780	39.23	71.59	190.45	110.82
	July	325,050	26	310	39.46	71.59	198.86	111.05
	August	162,520	26	1,500	39.55	71.59	203.11	111.14
	September	314,560*	26	2,890	39.67	71.59	208.06	111.26

\*Block I lignite reserves are depleted by approximately mid-September, 1983.

Table 3-4 Lignite Reserve Estimate, Mining Block 1

Cut No.	Total Length of Cut (m)	Average Depth of Overburden (m)	Average Lignite thickness in cut (m)	Volume of Lignite in cut (m <sup>3</sup> )
1	4,280	16.5	3.2	501,300
2	2,480	18.8	3.7	336,100
3	2,450	18.4	3.6	322,600
4	2,220	18.9	3.6	292,000
5	2,160	18.6	3.5	277,100
6	2,040	18.6	3.5	260,900
7	1,970	18.1	3.4	245,400
8	1,860	17.8	3.4	230,700
9	1,750	17.9	3.4	217,600
10	1,670	17.3	3.3	201,000
11	1,530	17.4	3.7	207,600
12	2,700	15.3	3.3	325,700
13	2,720	15.1	3.3	328,600
14	2,730	15.0	3.2	320,000
15	2,760	14.8	3.2	322,700
16	2,760	14.8	3.4	343,500
17	2,760	15.8	3.5	353,900
18	2,780	16.4	3.6	366,600
19	2,790	16.9	3.7	378,200
20	2,810	17.3	3.8	390,200
21	2,800	17.9	3.9	399,900
22	2,840	18.2	3.8	385,400
23	2,790	18.8	3.8	387,800
24	2,820	19.6	3.8	391,300
25	2,880	20.0	3.7	390,200
26	2,910	20.5	3.7	393,200
TOTAL LIGNITE RESERVE				8,579,500

Table 3-5 Lignite Reserve Estimate, Coronach Mine

Mining Block No.	Area of Mining Block (hm <sup>2</sup> )	Average Depth of Overburden (m)	Average Lignite Thickness (m)	Lignite Reserve (m <sup>3</sup> )
1	248.6	17	3.5	8,579,500
2S	457.6	30	2.9	13,116,500
2N	267.0	31	3.7	9,766,500
3S	425.1	27	3.4	14,253,600
3N	183.9	32	3.0	5,604,000
4S	431.8	35	3.6	15,528,500
4N	256.3	36	3.3	8,514,400
5S	412.4	22	3.4	14,202,200
5N	242.1	34	3.7	9,077,800
6S	372.1	30	3.4	12,816,000
6N	257.5	31	4.2	10,907,900
7S	328.7	30	3.8	12,621,600
7N	300.6	34	4.3	12,827,300
TOTALS	4,183.9			147,815,800

Average lignite thickness throughout mine = 3.5 m.

### 3.3 Reclamation Plan

The reclamation effort at the Coronach mine will run concurrently with the mining operation (Nelson et al, August, 1977). Four possible levels of reclamation originally were envisioned. In order of increasing cost and complexity they are: (1) natural reclamation, (2) reclamation to wildlife habitat, (3) reclamation to livestock grazing, and (4) reclamation to cereal crop production.

The stated objective of reclamation as presently planned is to return most of the mined area to cereal crop production. Final cuts in some mining blocks will remain unfilled, but most of the area is to be reclaimed for agricultural production.

### 3.4 Mining Equipment and Facilities

#### 3.4.1 Mining Equipment

A Bucyrus-Erie 2570-W walking dragline with  $69 \text{ m}^3$  ( $90 \text{ yd}^3$ ) bucket will be used to remove overburden at the Coronach mine. Photos 3-3 and 3-4 show this dragline being assembled at the SPC Coronach mine. A second dragline will be purchased at a later date so that mining in the northern tier of blocks may proceed at the same time the blocks south of Girard Creek are being mined.

Lignite will be excavated from the Hart Seam by a Bucyrus-Erie 195-B power shovel. Additional units will be purchased at a later time to meet the demands of mining two blocks simultaneously.

A fleet of 136-tonne (150-ton) haulage trucks to transport lignite from the pit to the loadout facility have been purchased. In addition, a 600-horsepower articulated dozer and a  $16 \text{ yd}^3$  front-end loader will be used to clean the lignite and assist with loading operations in the pit. This and additional mine support equipment is listed in Annex A.

#### 3.4.2 Loadout Facilities

Figure 3-2 shows a layout of the mine loadout facilities. Trucks hauling from the pit will unload at the 590-tonne (650-ton) capacity truck dump hopper where the lignite will be passed through a grizzly and into a 1,090 tonne/hr (1,200 ton/hr) coal breaker. A conveyor will carry lignite from the breaker to the 13,000 tonne (14,300-ton) loadout stockpile for loading into cars of the coal train.

#### 3.4.3 Maintenance Shop and Fueling Area

Also located at the loadout facility are the mine services building, locomotive storage shed, equipment storage shed, loading docks, cold oil storage shed, fuel storage tanks, refueling area, and truck parking aprons. These features all are shown in Figure 3-2.

PHOTO 3-3 COMMITTEE MEMBER D. KIMBALL STANDING INSIDE 69 m<sup>3</sup> (90 yd<sup>3</sup>) BUCKET OF THE BUCYRUS ERIE 2570W DRAGLINE AT THE SPC MINE NEAR CORONACH, SASK.



PHOTO 3-4 BUCYRUS ERIE 2570W WALKING DRAGLINE UNDER CONSTRUCTION AT THE SPC CORONACH MINE SITE NEAR CORONACH, SASKATCHEWAN



### 3.5 Mining Procedures

The following several paragraphs detail the work performed in advance of actual mining, the mining operation itself, and the reclamation procedures following mining.

#### 3.5.1 Drilling and Sampling

Prior to mining, it is necessary to determine the thickness of overburden and lignite in the area to be mined. Exploratory holes drilled on wide centers cannot provide the detailed information needed to plan for mining and production rates; therefore, additional drilling is carried out at spacings of 30 to 150 m (100 to 500 ft). In addition to depth information, lignite samples are obtained and are subjected to a chemical analysis to determine the quality of the lignite which will be mined.

Drilling and sampling are carried out throughout the life of the mining operation.

#### 3.5.2 Dewatering

Because the Hart Seam is an aquifer and because other aquifers exist in the overburden, it is necessary to drain this excess water from above and from within the lignite seam in order to carry out the mining operation. In addition, stream drainages will be interrupted by the mining process and it will be necessary to divert natural streams to prevent their entering the active pit. Precipitation and runoff will also enter the pit directly and must be removed for mining to continue.

##### (a) Groundwater

In advance of the mining operations, wells are drilled to a depth slightly below the base of the Hart Seam. Casing is set and pumps are installed. Water entering the wells from aquifers above and within the lignite then is pumped from these wells and is discharged into Girard Creek. This pumping withdraws water from the aquifers more rapidly than recharge can occur, and a cone of depression results. Mining then is carried out in the area of the cone of depression.

##### (b) Surface Water

Natural drainages that are interrupted by the mine must be diverted so that excess water does not enter the pit and impede mining. In order to do this at the Coronach mine, areas will be enlarged in the natural drainage courses near the pit boundary. These enlargements, capable of holding a few hundred thousand cubic meters of water, will be pumped as needed, and will discharge into 150 mm (6-in) or 200 mm (8-in) irrigation pipe to be routed to a settling pond and into Girard Creek.

(c)

### Mine Pit and Sump Drainage

Even after dewatering wells depress the water table in the area to be mined, a small amount of water associated with perched aquifers will empty into the excavation when the area is disturbed. In addition, direct precipitation into the pit area and on unreclaimed spoil piles will result in an accumulation of water in the sump of the mine pit.

A float will be used to support pumps in the mine sump. These pumps will drain the mine sump, discharging through a settling pond or stilling barrel into Girard Creek. The pumping rate will be adjusted to meet the conditions extant in the pit.

#### 3.5.3 Topsoil Removal and Storage

Initially, the mining rate at the Coronach mine site will be low enough during the stockpiling of a 2-unit 30-day lignite supply and during the startup and checkout of the first 300-MW unit that the 2570-W dragline will be under-utilized.

In order to get more use out of this machine, it is anticipated that the dragline will be used to transfer topsoil piled by dozers in advance of the mining operation to leveled spoil piles behind the working cut. This method is not presently used elsewhere and may or may not prove feasible at the Coronach mine site.

Conventional means of topsoil removal and transportation around the working cut will be employed if use of the dragline for topsoil transfer proves unsatisfactory. This method relies on 27 m<sup>3</sup> (35 yd<sup>3</sup>) pan scrapers to strip topsoil to the desired depth, haul it around the working cut, and dump it in stockpile areas or spread it over leveled spoil piles undergoing reclamation.

Regardless of which method is used, topsoil removal will be possible only during the frost-free months of the year, so advanced stripping of topsoil will be necessary in order to continue mining lignite during the winter months.

Reclamation efforts will commence on spoil piles 3 or 4 cuts behind active mining, therefore a considerable quantity of topsoil will have to be stockpiled at any given time. This soil will be seeded with grass to help control weeds and to minimize erosion losses (Nelson et al., August, 1977).

Because of the anticipated size the mining blocks at the Coronach mine, no topsoil should remain in a stockpile for more than 3 to 5 years before being reused in reclamation.

### 3.5.4 Excavation of Overburden and Spoil Placement

After topsoil has been stripped ahead of a mining cut, the overburden above the lignite must be removed. At the Coronach mine site, this material consists of glacial till, clays, silts, and sands. The material is readily ripped with dozers and may be excavated with a dragline bucket. No hard rock or resistant formations requiring blasting occur above the lignite in this area. (Some blasting will take place at the mine site during maintenance of the dragline bucket since dynamite must be used to loosen worn teeth for replacement.)

A dragline will excavate overburden with a  $69\text{ m}^3$  ( $90\text{ yd}^3$ ) bucket (see Photo 3-3) and will transfer the material across the turnover cut to spoil areas where the material will be dumped. The placement of this spoil is dependent upon the type of overburden encountered in mining.

#### (a) Favorable Overburden

Alluvium, glacial till, and the clays, silts, and sands of the Upper Ravenscrag Formation not containing alkali levels that would adversely affect plant growth are considered favorable overburden. These materials require a minimum of handling and may simply be removed from the advancing high wall and placed in spoil piles behind the active cut as shown in Figure 3-6.

#### (b) Unfavorable Overburden

Where alluvium, tills, or bedrock clays, silts, or sands contain saline deposits that would inhibit plant growth and render reclamation efforts difficult or impossible, a means of selective spoil placement will be used (Nelson et al, August, 1977). This process is shown in Figure 3-7.

Initial stripping in areas of unfavorable overburden will stockpile any unfavorable material prior to excavation and placement of the unfavorable material. Favorable overburden then will be placed over the unfavorable material and reclamation will proceed as in areas of favorable overburden.

This process requires skilled dragline operators and slows the mining operation because of the extra overburden handling steps. It also raises the question of finding and delimiting areas of favorable overburden in otherwise unsatisfactory materials. Selective placement appears necessary in order to bury the alkaline soils known to exist in several of the mining blocks north of Girard Creek. Reclamation efforts in these blocks are much less likely to achieve acceptable results if selective spoil placement procedures are not employed.

### 3.5.5 Lignite Mining Operations

#### (a) Recovery of Lignite from the Hart Seam

FIGURE 3-6. Schematic Diagram of Mining Procedure at Coronach Mine

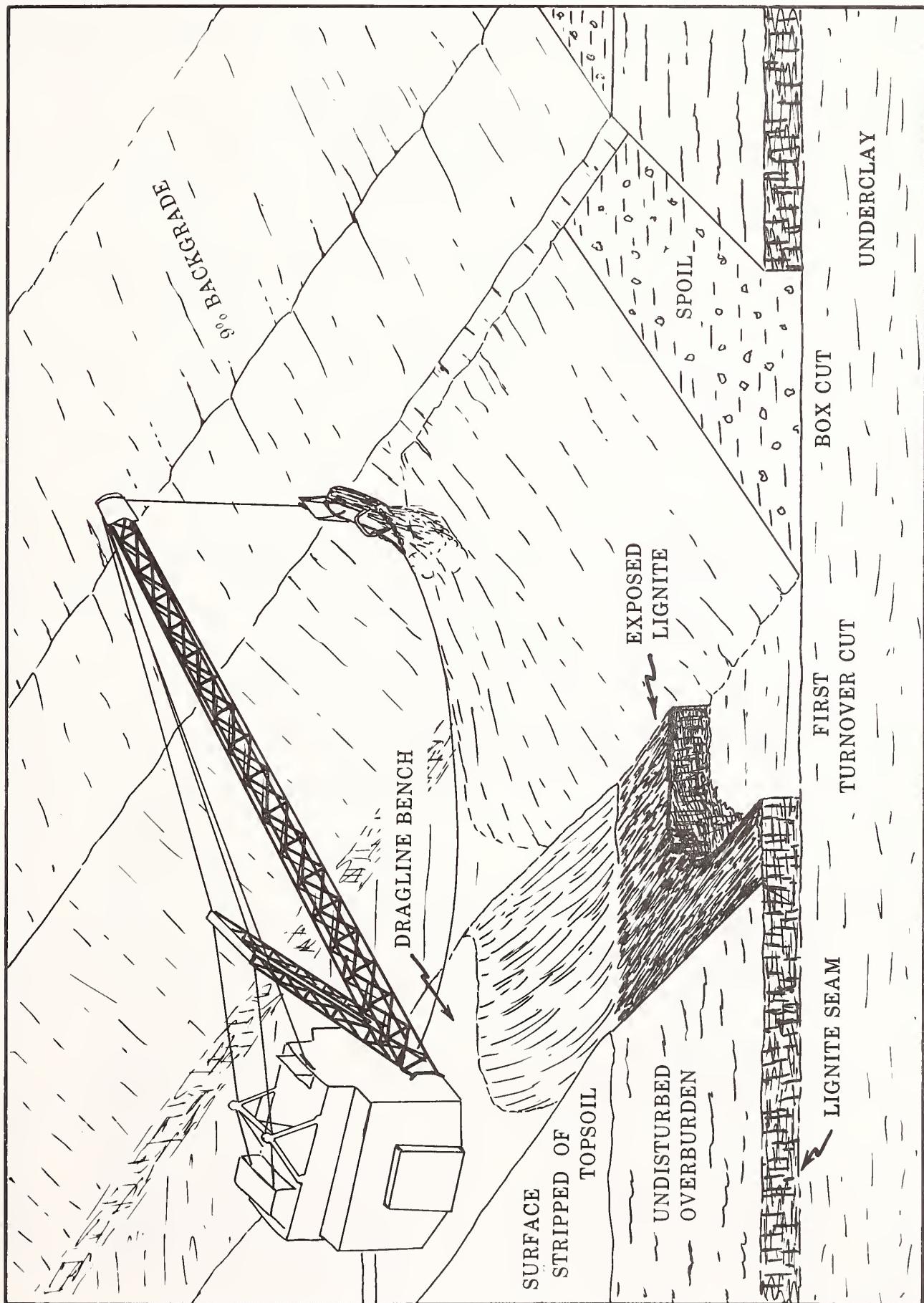
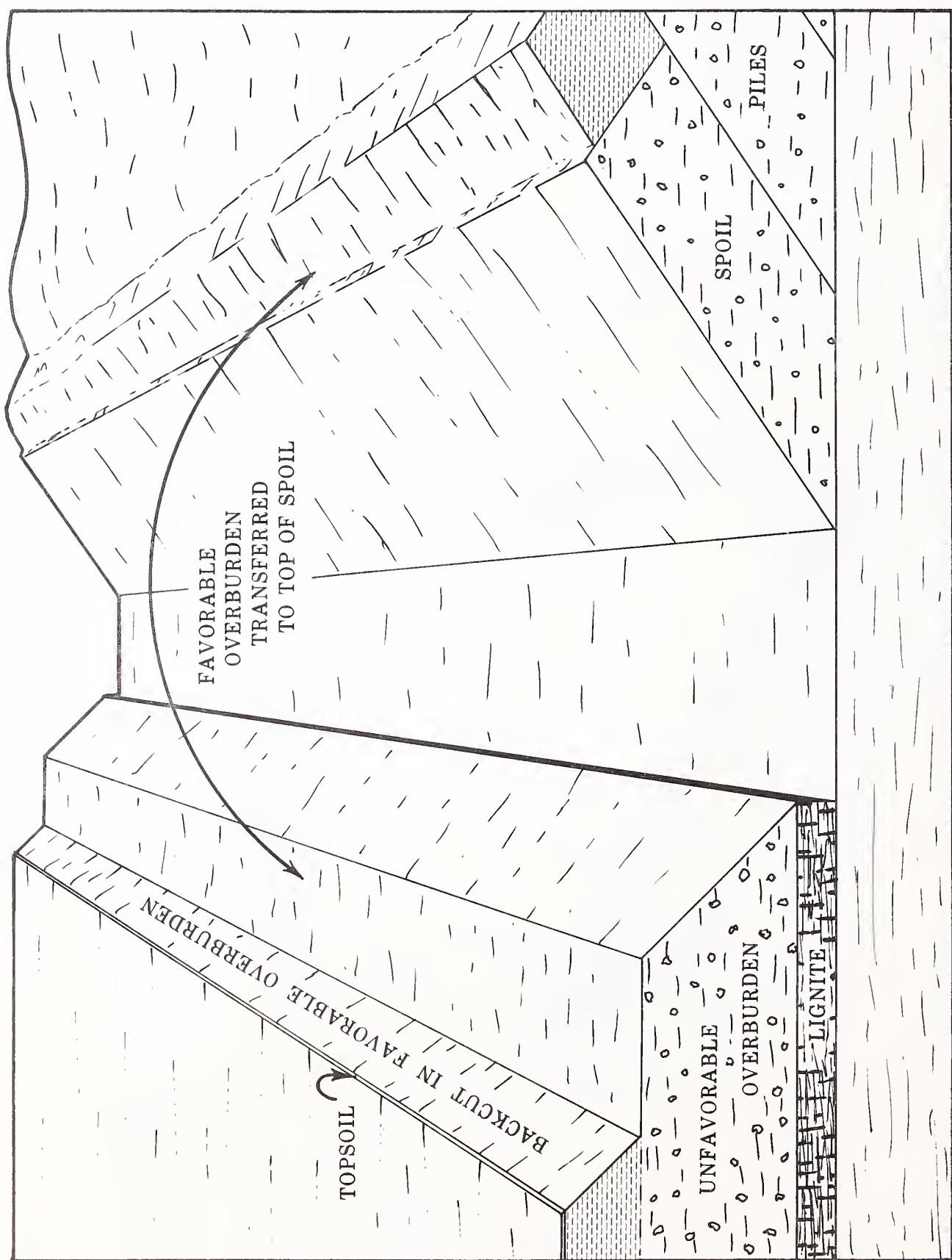


FIGURE 3-7. Schematic Diagram of Selective Spoil Placement



The dragline will remove overburden within a few centimeters of the top of the lignite seam. Cleanup operations then will be performed by dozers and front-end loaders to minimize the amount of waste rock that is mined along with the lignite.

Dozers are used to rip the lignite and excavation of the seam is performed with a power shovel. This procedure will leave only a few centimeters of lignite at the bottom of the seam and front-end loaders will be used to recover some of this material.

By using these procedures, most of the lignite in the seam will be recovered and will be suitable for fueling the boilers at the Poplar River Power Station. During the initial phases of mining in Block 1, it is anticipated that only about 80 percent of the available lignite will be recovered. This will be due to inexperienced equipment operators and to the presence of partings in the lignite. An overall lignite recovery factor of 85 percent probably will be achieved and this factor could rise even higher during the life of the mine, but for the purposes of this study, an 85 percent recovery factor has been assumed.

(b) Loading and Haulage

The power shovel will be the primary means of loading lignite into 136-tonne (150-ton) haul trucks. Front-end loaders will help load the haul trucks and will keep the pit bottom free of spilled lignite.

The trucks then will climb haulage roads out of the pit bottom to the lignite loadout facility where they will dump their payload. Because the working cut is continually advancing, haulage roads and ramps out of the pit also are continually changing. These roads are designed to make the shortest haul from the pit bottom to the loadout facility. The practical haul limit for effective use of 136-tonne trucks is about 8 km (5 mi).

(c) Processing and Shipment

When the lignite is first removed from the seam, it will be wet and very blocky, with pieces as large as 1 m (3 ft) in diameter. This material will be broken up during the loading into trucks and dumping at the loadout facility, resulting in pieces generally less than 0.3 m (1 ft) in size.

At the loadout facility, the lignite will be passed through a grizzly and into a breaker where it will be reduced to less than 80 mm (3 in) in size. This size reduction will facilitate handling and air drying of the lignite.

From the breaker, the lignite will be transported by conveyor to a large loadout stockpile from which it will be loaded into 82 tonne (90-ton) railroad hopper cars and transported to the power station.

### 3.5.6      Reclamation

Reclamation activities at the Coronach mine site are expected to proceed by leveling spoil, replacing topsoil, and reseeding.

#### (a)      Desired Level of Reclamation

The reclamation effort at the Coronach mine will run concurrently with the mining operation. Four possible levels of reclamation originally were envisioned. In order of increasing cost and complexity they are: (1) natural reclamation, (2) reclamation to wildlife habitat, (3) reclamation to livestock grazing, and (4) reclamation to cereal crop production.

The stated objective of reclamation as presently planned is to return most of the mined area to cereal crop production. Final cuts in some mining blocks will remain unfilled, but most of the area is to be reclaimed for agricultural production.

#### (b)      Contouring and Leveling Spoil

Reclamation efforts are planned to follow mining by 3 or 4 spoil ridges. This will allow time for the spoil to compact and stabilize somewhat before reclamation commences.

Large dozer-bladed crawler tractors will be used to push spoil from the ridge peaks into the valleys between spoil banks and to smooth the surface to approximate final contours as shown in Figure 3-8. The maximum grade presently anticipated for reclaimed land is 9 percent.

#### (c)      Topsoil Placement

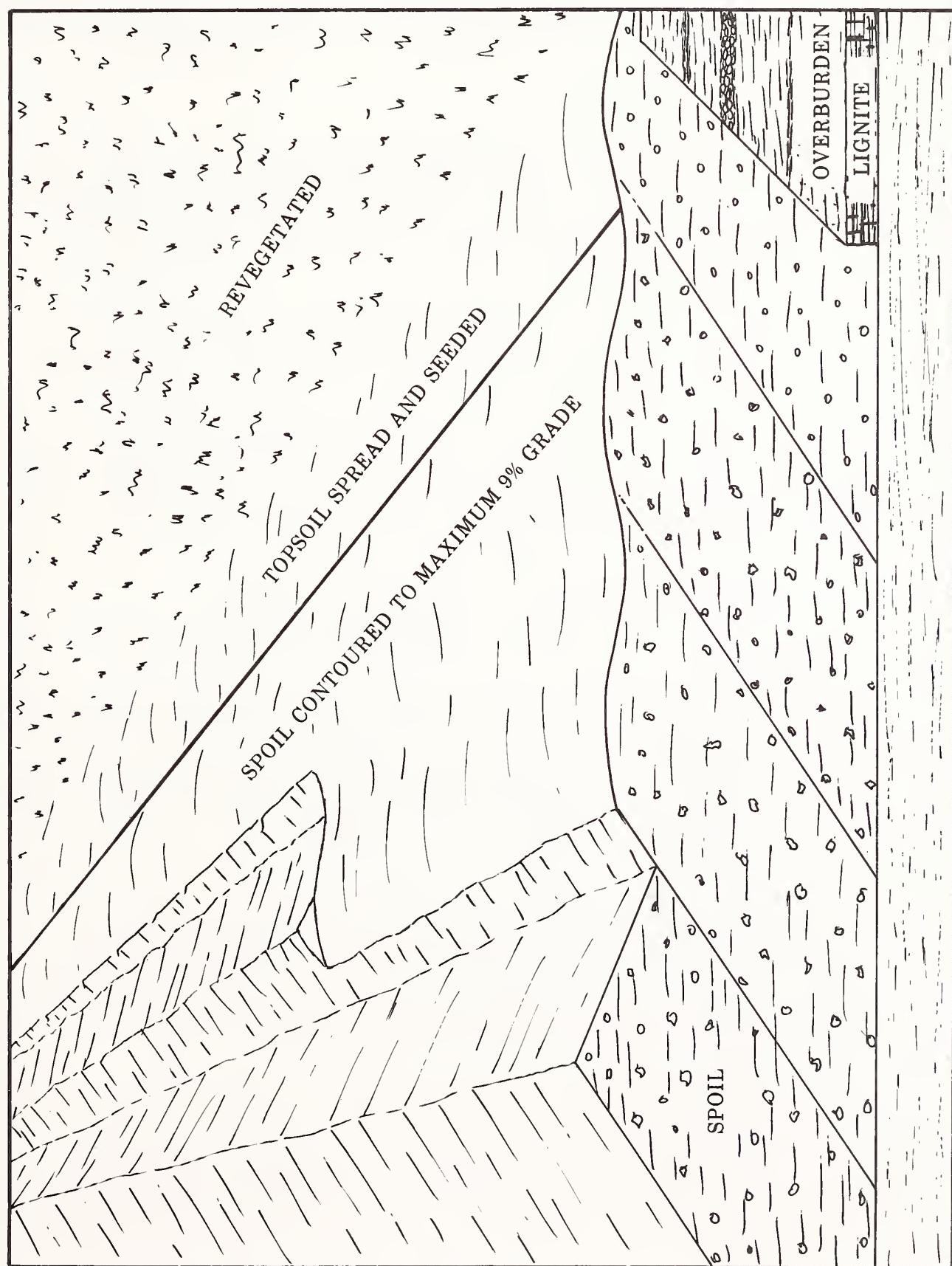
After leveling and contouring, topsoil will be hauled to the reclaim area and spread on the contoured spoil. Several combinations of machines could be used for this task, but scrapers appear to be the most efficient means of transporting and spreading topsoil and hence, are likely to be used for this purpose.

#### (d)      Reseeding

The next step in the reclamation process will be to fortify the freshly spread topsoil with fertilizer. The land surface then will be harrowed and planted with spring grain, fall rye, or winter wheat, depending on the time of year. These fast-germinating and rapid-growing cereal grains will provide initial cover to control erosion and weed encroachment.

Where insufficient rainfall or lack of soil moisture endangers the reclamation effort, supplemental irrigation probably will be required to help re-establish vegetation on the reclaimed lands.

FIGURE 3-8. Schematic Diagram of Spoil Reclamation at the Coronach Mine



Following the successful establishment of first-growth on the reclaimed land, a regular cropping program will be initiated to rebuild the soils over a period of several years, progressing through forage crops to cereal grain production.

(e) Reclamation Time Schedule

As presently planned, topsoil will be stockpiled in advance of the mining operation to preclude loss of this material during overburden removal. In addition, subsoils may be retained or selectively placed to cover less favorable overburden material in spoil piles remaining after lignite removal has taken place.

Mine spoil will be graded to maximum 9 percent slopes approximately one year after mining has taken place. The newly graded spoil will have topsoil spread over it and will be seeded in fall rye or other fast-germinating grain.

Two years after mining, grass will be sown and harvested on the reclaimed areas. Grass will continue to be grown on the reclaimed land for an additional three to five years before a cereal crop is sown and a regular schedule of cereal-grass rotation is established.

3.6 Mining Inputs and Losses to Surface Water Systems

3.6.1 Dewatering

In order to dewater the Coronach mine areas so that mining may be carried out, 12 wells have been drilled and completed in the Hart Seam. These wells presently yield an average flow of approximately 11,300 m<sup>3</sup> or about 9.1 ac-ft daily (3,340 ac-ft annually). This represents an input to surface water and a loss to groundwater. Water quality for this volume of flow is summarized in Table 3-6.

The volume of water pumped from dewatering wells is expected to decline over a period of several years as the cone of depression is stabilized. This steady-state water yield has been estimated (Mollard, 1976) at about 1,350,000 m<sup>3</sup> (1,100 ac-ft) annually. More authoritative estimates of longterm dewatering yield have been prepared by the Groundwater Committee. These rates are substantially greater than previous estimates, but the water quality is expected to remain the same.

An interim approval by the Saskatchewan Department of the Environment specifies that the level of suspended solids in Girard Creek not be increased by more than 10 mg/l and turbidity by more than 25 turbidity units over background as a result of dewatering. Also, B concentration is not to exceed 2 mg/l.

Table 3-6 Hart Seam Dewatering Well Water Quality Data  
(Values in mg/l except pH; based on data provided in Annex C)

<u>Parameter</u>	<u>Concentration</u>
pH	7.4
Total Dissolved Solids (TDS)	1,096
Total Alkalinity (as CaCO <sub>3</sub> )	611
Total Hardness (as CaCO <sub>3</sub> )	471
Calcium	98
Magnesium	55
Sodium	199
Potassium	6
Boron	1.8
Iron	1.4
Manganese	0.32
Carbonate (as CO <sub>3</sub> <sup>=</sup> )	nil
Bicarbonate (as HCO <sub>3</sub> <sup>-</sup> )	725
Sulfate	285
Nitrate	2.3
Chloride	4
Fluoride	0.30

### 3.6.2 Mine Pit and Sump Drainage

Even after dewatering, there will be some seepage into the active pit as well as precipitation directly into the pit and spoil pile runoff that will drain into the pit. The amount of seepage into the pit will be strongly dependent on the rainfall, overburden characteristics, and pumping rates of dewatering wells. There are presently no reliable data to indicate precisely what seepage rates will be encountered in the mining operation; however, an estimate of the seepage rate has been prepared from data gathered at Test Pit #3 by the Saskatchewan Power Corporation.

In Test Pit #3, water was observed flowing from the area of the walls, but not from the underclay beneath the lignite seam. The test pit wall area of approximately 3,500 m<sup>2</sup> (38,000 ft<sup>2</sup>) was observed to yield about 42,600 l/day (11,250 U.S. gpd). This gives us an average seepage rate of 2.47 l/day/m<sup>2</sup> (0.296 U.S.gpd/ft<sup>2</sup>) of wall area.

Seepage into spoil will be lower than into the pit itself, and it is anticipated that as reclamation proceeds, little, if any, contribution to water in the pit bottom will derive from this material. Thus, as the mine pit and unreclaimed spoil areas initially expand, more seepage will be encountered until a relatively steady condition obtains. This has been approximated in Table 3-7, but represents only a "best guess" at this time.

TABLE 3-7 MONTHLY SEEPAGE VOLUME INTO MINE PIT

YEAR	MONTH	REQUIRED LIGNITE VOLUME (m <sup>3</sup> )	CUT NO.	SEEING WALL AREA OF PIT (hm <sup>3</sup> )	MONTHLY VOLUME OF SEEPAGE (m <sup>3</sup> )
1979	April	51,650	1	2.48	8,600
	May	51,650	1	4.79	17,200
	June	51,650	1	7.11	24,700
	July	51,650	1	9.42	33,800
	August	51,560	1	11.74	42,200
	September	51,560	1	14.05	48,800
	October	51,560	1	16.37	58,800
	November	45,200	1	18.40	64,000
	December	46,700	1	20.49	73,600
1980	January	46,700	1	22.58	84,100
	February	43,690	2	22.16	74,500
	March	162,520	2	19.40	69,700
	April	157,280	3	17.18	59,700
	May	162,520	3	17.18	61,700
	June	0	3	17.2	59,700
	July	162,520	4	17.2	61,700
	August	162,520	4	17.2	61,700
	September	157,280	5	17.2	59,700
	October	162,520	5	17.2	61,700
	November	157,280	6	17.2	59,700
	December	162,520	7	17.2	61,700
1981	January	162,520	7	17.2	61,700
	February	146,800	8	17.2	55,800
	March	162,520	8	16.6	61,700
	April	157,280	9	16.6	57,600
	May	162,520	10	16.0	57,500
	June	0	10	16.0	55,600
	July	162,520	11	15.4	55,300
	August	162,520	12	14.8	53,100
	September	157,280	12	14.2	49,400
	October	162,520	13	13.6	48,800
	November	157,280	13	13.6	47,300
	December	162,520	14	13.6	48,800
1982	January	162,520	14	13.6	48,800
	February	146,800	14	13.6	44,100
	March	162,520	15	13.6	48,800
	April	157,280	15	13.6	47,300
	May	162,520	16	13.6	48,800
	June	45,200	16	13.6	47,300
	July	209,230	17	13.6	48,800
	August	209,230	17	13.6	48,800
	September	202,480	18	13.6	47,300
	October	325,050	19	13.6	48,800
	November	314,560	20	13.6	47,300
	December	325,050	20	13.6	48,800
1983	January	325,050	21	13.6	48,800
	February	293,590	22	13.6	44,100
	March	325,050	23	13.6	48,800
	April	314,560	24	13.6	47,300
	May	325,050	24	13.6	48,800
	June	157,280	25	13.6	47,300
	July	325,050	26	13.6	48,800
	August	162,520	26	13.6	48,800
	September	314,560	26	13.6	47,300

Each operating pit at the Coronach mine will eventually total approximatley 100 ha (360 ac) of open cut and unreclaimed spoil. This is the estimated area that will intercept precipitation and drain into the mine pit. The annual precipitation in this area averages about 373 mm (14.7 in). It is assumed that approximately 80 percent of the annual precipitation falling onto the disturbed area of pit and spoil is lost to evaporation and recharge of groundwater and that 20 percent shows up in the mine sump (a very conservative estimate). This yields an annual volume of about 74,600 m<sup>3</sup> (60 ac-ft).

Combined with the average annual seepage inflow of 598,300 m<sup>3</sup> (485 ac-ft), this yields an average annual value of 672,900 m<sup>3</sup> (545 ac-ft) of pit and sump drainage which is an input to surface water and a loss to groundwater. The quality of this water is not well characterized, but analysis of 13 water samples from Test Pit #2 are summarized in Table 3-8.

Table 3-8      Test Pit #2 Water Quality Data

(Values in mg/l except pH; Based on data provided in Annex C)

<u>Parameter</u>	<u>Concentration</u>
pH	7.6
Total Dissolved Solids (TDS)	564
Total Alkalinity (as CaCO <sub>3</sub> )	385
Total Hardness (as CaCO <sub>3</sub> )	395
Calcium	81
Magnesium	47
Sodium	46
Potassium	8
Boron	0.8
Iron	1.1
Manganese	0.23
Carbonate (as CO <sub>3</sub> <sup>=</sup> )	nil
Bicarbonate (as HCO <sub>3</sub> <sup>-</sup> )	318
Sulfate	103
Nitrate	18
Chloride	<5
Fluoride	0.31

These data indicate that mine sump water will be of generally good quality. However, increased TDS levels might be expected due to the mining activity.

### 3.6.3 Diversion Around Mine Site

Mining activity ultimately will affect eight distinct drainage areas on the south side of Girard Creek and four more areas on the north side of the creek. These range in size from 2.9 to 29 km<sup>2</sup>.

Water diversion around the mine site will be accomplished by directing the basin channel into a sump or natural depression and then pumping the collected water around the mine area and into Girard Creek. The quantities of water so intercepted and diverted will be dependent upon the location and extent of the mining activity and upon the annual precipitation; however, an estimate can be made by determining the drainage area for which the mining activity will intercept overland flow and ephemeral streams and applying the runoff rates of Table 3-9. For example, in Mining Block 1, the interception area is 16.5 km<sup>2</sup> and the average annual quantity of water to be diverted will be 343,200 m<sup>3</sup>.

Because this water is merely being re-routed around the site of active mining, it is not expected to result in any significant gains or losses to Girard Creek in comparison to pre-development conditions. Similarly, because the diverted water will not enter the mine pit, no significant changes in water quality are expected.

The loss to surface water and input to surface water are expected to be equal, so estimates of the quantity of water what will be diverted around the mine site are not considered important to this study. This assumes that certain conditions will be met in the design and operation of this system:

- (a) Sumps or depressions will be designed to hold the maximum quantities of predicted runoff with adequate settlement time following a 50-year 24-hour precipitation event before discharge.
- (b) Pumpout of storage areas will be accomplished by decanting the surface water after a sufficient period of time to permit settling of suspended solids.
- (c) As soon as sediment collection in storage or settling basins has decreased the capacity of the facility by 10 percent, the accumulated sediment will be removed from the basins.
- (d) Pumpout water will be returned either directly to Girard Creek or to other natural drainage channels.
- (e) If the diverted waters are returned to Girard Creek by open channels, design measures will assure that the channels will remain stable at

Table 3-9 Runoff Rate Estimates for the Coronach Mine Area<sup>1</sup>

Month	Runoff (mm)	Month	Runoff (mm)
January	0.0	July	2.7
February	0.0	August	2.5
March <sup>2</sup>	4.6	September	1.8
April	1.5	October	1.0
May	2.0	November	0.5
June	4.3	December	0.0
Total Annual Runoff = 20.9 mm			

<sup>1</sup> Based on average flow records and a drainage area of 737.1 km<sup>2</sup>, the annual discharge for the East Poplar River basin is 20,876 m<sup>3</sup>/km<sup>2</sup>. This is a runoff of 0.0209 m or 5.6 percent of the average annual precipitation of 0.374 m.

<sup>2</sup> This assumes that half of the November runoff and the December, January, February, and March runoff occur as spring runoff during the month of March.

Table 3-10 Statistical Summary of Precipitation Records for Swift Current, 46-Year Period from 1931-1976.

Month	Average Precipitation (mm)	Standard Deviation (mm)	Range
			Minimum (mm)      Maximum (mm)
January	20	9	3      42
February	17	10	2      45
March	16	11	3      53
April	26	18	1      81
May	36	24	5      113
June	77	43	12      216
July	49	32	5      152
August	45	31	2      130
September	32	24	0      86
October	18	14	1      64
November	18	12	1      54
December	19	10	0      45
TOTALS	373	35	1,081

Maximum annual precipitation (1954) 539 mm.

Minimum annual precipitation (1937) 206 mm.

predicted peak flows by lining the channels or by designing the ditches with similar lengths and gradients as former drainage channels that existed prior to mining.

- (f) Design channel capacity for open ditches will be capable of passing a minimum 25-year run-off event with some freeboard.

### 3.6.4 Non-Point Source Discharge from Reclaimed Areas

Total pre-mining runoff for all mining blocks is estimated to be 880,000 m<sup>3</sup>/year based on an average annual runoff of 0.021 m (see Tables 3-9 and 3-10). Pre-mining sediment yield from all mining blocks is approximately 4,125 m<sup>3</sup>/year, as estimated using a regional rate of approximately 98 m<sup>3</sup>/km<sup>2</sup>/year. Field observations made near Coronach in May, 1978 tend to confirm this relatively low rate of sediment loss from the drainage area.

Mining will disrupt a total of approximately 41.8 km<sup>2</sup> on either side of Girard Creek during the life of the project. Average annual surface area under reclamation will vary according to the size of the block being mined at any given time.

Post-mining runoff (Figure 3-9) will increase dramatically from the reclaimed surface. A multiplier of four times the pre-mining rate was used to estimate post-mining runoff from reclaimed areas. This increase is estimated to decrease at a linear rate of decay such that pre-mining rates are attained in the seventh year after reclamation commences. Annual runoff from the total mine area is estimated to peak at approximately 1,190,000 m<sup>3</sup> in 1990, an increase of nearly 35 percent over pre-mining annual runoff volumes.

Post-mining sediment yield (Figure 3-10) likewise is expected to increase significantly from the reclaimed surface. A multiplier of ten times the pre-mining rate was used to estimate post-mining sediment yield. This increase is expected to decrease at a near-linear rate such that pre-mining rates are attained in the seventh year after the commencement of reclamation. Annual sediment yield from the total mine area is estimated to peak at approximately 5,720 m<sup>3</sup> in 2014, an increase of 39 percent over pre-mining annual sediment yields. Actual sediment yields may, however, be substantially greater than those estimated here.

The level of reclamation has not yet been definitely determined, so predictions of runoff quantity and sediment yield have been based on the presumption that the disturbed mine area will be returned to cereal crop production as indicated by the Saskatchewan Power Corporation. It is believed the intensive land management practices common to cereal crop production will provide necessary control of excess runoff and sediment yield.

The following conditions also are inherent in the assumption of reclamation to cereal crop production (and the estimates shown in Figures 3-9 and 3-10):

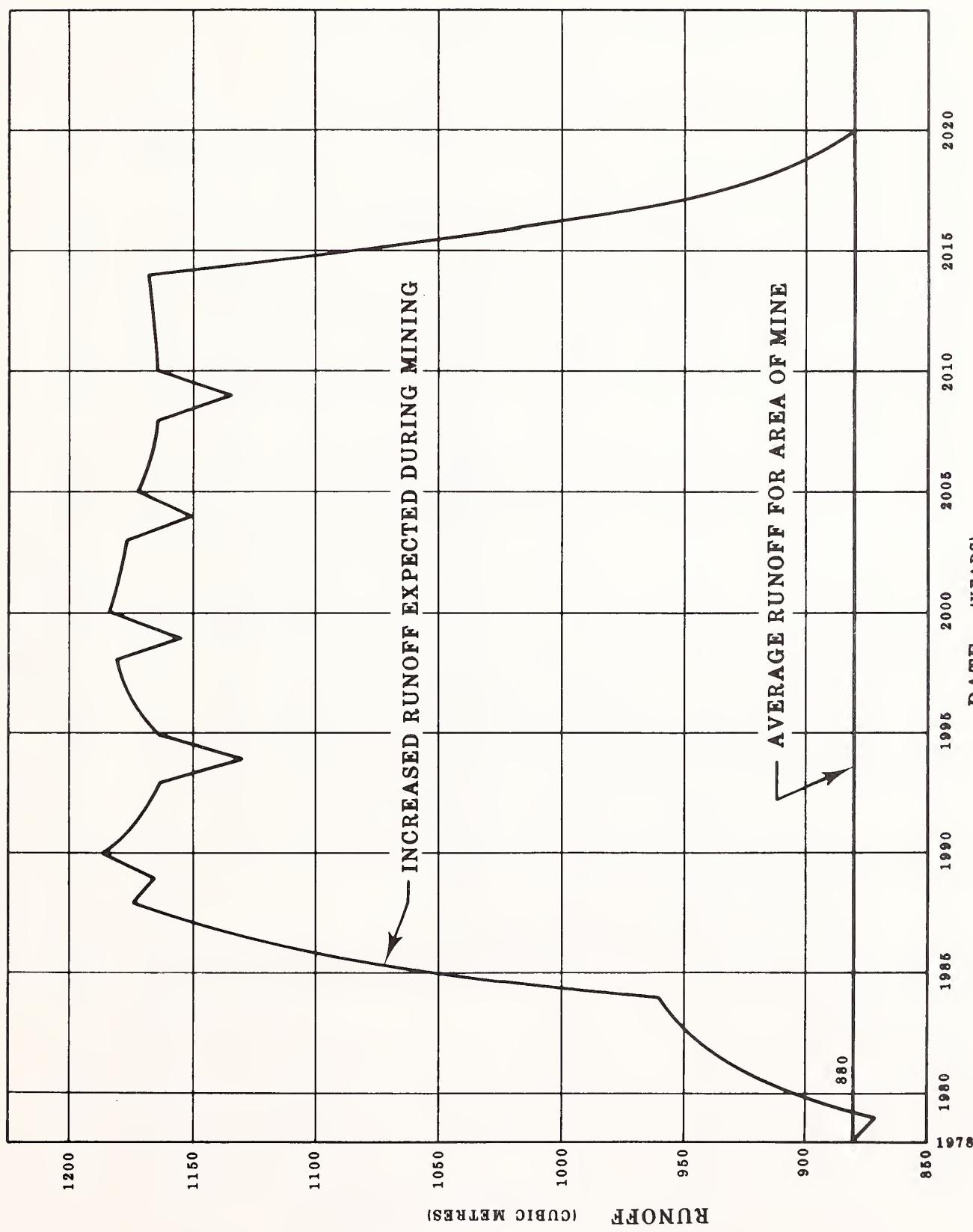


Figure 3-9. Runoff from mine area.

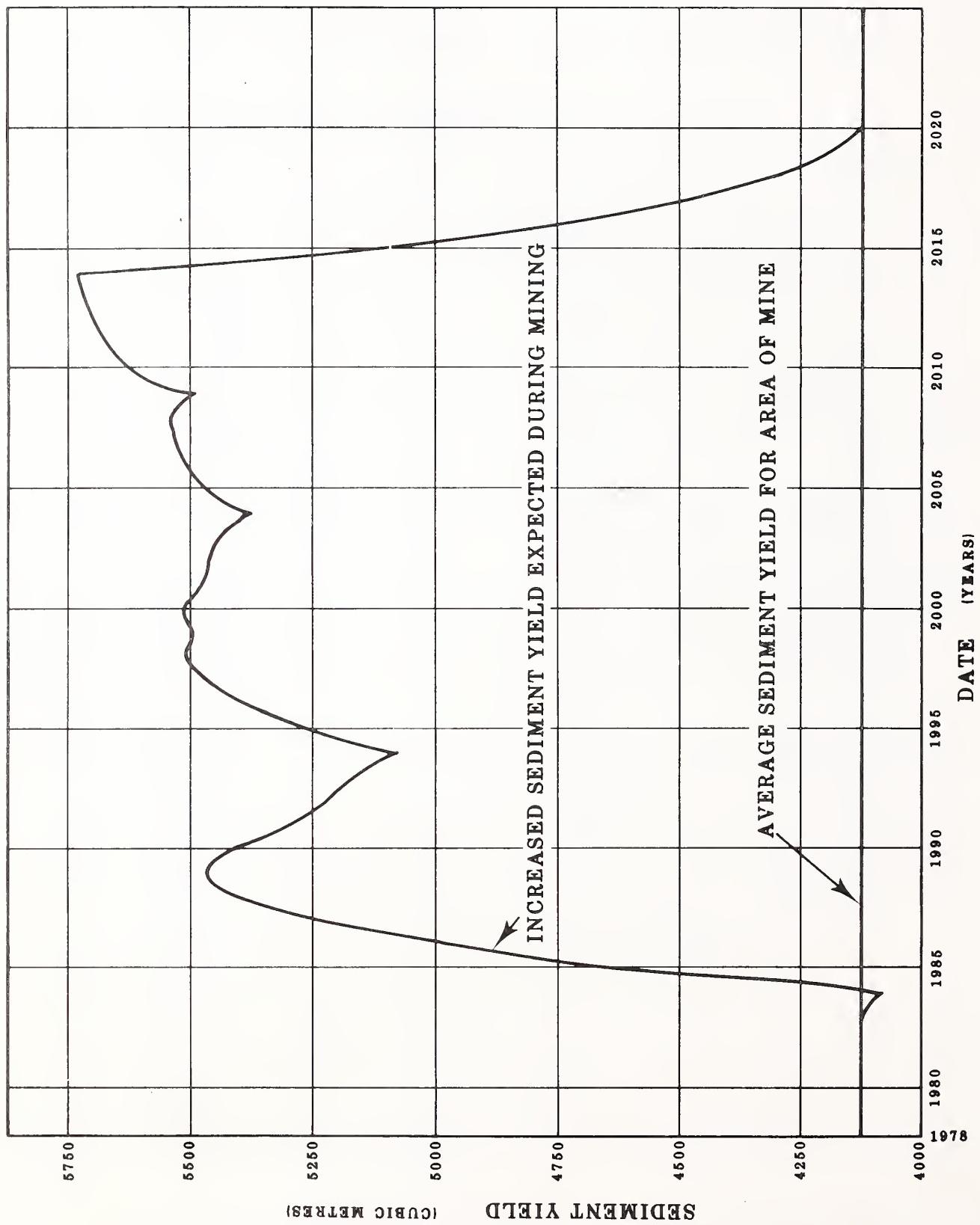


Figure 3-10. Estimated sediment yield from Coronach mine.

- (a) The post-mining reclaimed surface will drain into Girard Creek with no substantial closed depressions.
- (b) The post-mining surface will be returned to cereal crop production as soon as practicable following levelling and shaping of spoils and the replacement of stockpiled topsoil
- (c) Crop production operations will utilize all intensive agricultural management practices necessary to promote infiltration, reduce runoff and overland flow, and retard erosion.
- (d) Because mining south of Girard Creek will take place from south to north (down gradient), sediment and runoff produced from the surface undergoing reclamation will be diverted around the mining operations. Such diversions will trap sediment, but will pass all runoff. Following the completion of mining, the drainage will be restored and routed through the final cut (unless it is back-filled). Thus, the final cut will act as a sediment trap and keep excess sediment from entering Girard Creek.
- (e) Mining north of Girard Creek also will take place from south to north (up gradient in this case), and will include re-integration of the post-mining surface with the Girard Creek drainage.

### 3.6.5 Sanitary and Other Wastes

#### (a) Sanitary Wastes

Sanitary wastes at the mine site will originate from showers and sanitary facilities in the mine services building, sanitary facilities in the equipment storage shed, and equipment washing.

Cambrian Engineering, Ltd., was engaged to design the wastewater handling system at the mine site. A two cell lagoon system was chosen and designed using the following criteria:

(1) Equipment washwater	13,600 l/day (3,000 Igpd)
(2) Sanitary 70 employees in 24-hour period 70 gallons per capita per day (includes one shower per day)	22,300 l/day (4,900 Igpd)
(3) Water treatment filter backwash	2,270 l/day (500 Igpd)
Total	38,200 l/day (8,400 Igpd)

- (4) Average BOD loading assuming a strength of 200 ppm equals 7.6 kg/day (16.8 lb/day) although BOD probably will be much lower because the majority of water is wash and shower water.
- (5) First cell at 33.6 kg BOD/ha/day (30.0 lb BOD/ac/day) = 0.23 ha or 45 m by 50 m.
- (6) Second cell with a 180-day storage capacity of 7,155 m<sup>3</sup>, dimensions of 86 m by 43 m at 2 m depth and 3:1 side slopes.

The Saskatchewan Department of the Environment has approved the construction of the lagoon system but not its operation. The Department has stated that point source discharges of domestic wastewaters will not be allowed to enter Cookson reservoir, and have suggested that the SPC look into an effluent irrigation scheme. The Department will not allow the lagoon effluent to discharge directly into Girard Creek and hopes that the company will develop a means to use the lagoon effluent as irrigation water to revegetate reclaimed land.

It would seem safe to assume that the sanitary wastewater and vehicle washwater at the mine equipment maintenance and coal transfer area will not significantly alter the quality and quantity of water in Cookson Reservoir.

(b) Mine Shop and Maintenance Wastes

The oil and fuel storage area will be surrounded by a 1 m (3 ft) dike constructed of compacted clay. The dike area will have a capacity of about 205 m<sup>3</sup> (7,240 ft<sup>3</sup>) (greater than the capacity of all storage tanks).

All used oil and lubricants will be collected and stored in a used-oil tank (inside the diked area). The SPC is looking into using this oil either as an additive to the coal or as a supplement to diesel fuel. The company feels that they will add the oil to the coal to be burned in the power plant.

Salvageable scrap metal will be stored and sold as scrap. Unsalvageable metal will be buried in the bottom of the mined areas and covered with overburden. Other solid wastes would include garbage, paper, wood, and wastes from the machine shop. This material will be disposed of in a modified landfill. The landfill site is not yet established, but the site must be approved by the Department of Health and the Department of the Environment prior to development. Environment Saskatchewan will require that the site is located and operated such that ground and surface waters are not contaminated.

3.7        Mining Inputs and Losses to Groundwater System

3.7.1      Dewatering

The dewatering of overburden and lignite will be a major loss to groundwater in the Coronach mine area. As previously determined, this presently amounts to about 4,120,000 m<sup>3</sup> (3,340 ac-ft) annually and may increase during a period of several years.

3.7.2      Mine Pit and Sump Drainage

As previously discussed, the seepage into the pit and precipitation into the mine area represent a loss to ground-water of about 672,900 m<sup>3</sup> (545 ac-ft) annually.

3.7.3      Leachates from Spoil Materials and from Reclaimed Land

A study (Pagenkopf, May 17, 1978) of the leachates from mine spoil materials in the Coronach mine area was commissioned by the Plant, Mine and Reservoir Operations Committee.

In this study, samples of overburden obtained from exploratory drilling were leached with distilled water and the weight loss was reported as readily soluble salts. The leachate solution then was analyzed to determine the type and concentration of dissolved salts leached from the overburden. The results are presented in detail in Annex D and summarized in Tables 3-11 and 3-12.

Table 3-11            Readily Soluble Salts in Overburden

<u>Sample Number</u>	<u>Weight Loss kg/m<sup>3</sup></u>	<u>Sample Number</u>	<u>Weight Loss kg/m<sup>3</sup></u>
1	0.75	19	6.37
2	11.28	20	1.82
3	1.50	21	5.18
4	1.29	22	2.31
5	7.08	23	7.50
6	1.96	24	2.47
7	0.93	25	4.97
8	0.90	26	5.78
9	1.07	27	2.08
10	1.28	28	2.34
11	2.54	29	1.03
12	4.90	30	1.83
13	1.23	31	4.57
14	6.26	32	3.47
15	2.77	33	4.52
16	1.01	34	7.35
17	3.26	35	3.73
18	0.89	36	5.63

TABLE 3-12. Chemical Analyses of Leachate Solutions

SAMPLE NUMBER	HCO <sub>3</sub> <sup>-</sup> mg/1	CO <sub>3</sub> <sup>=</sup> mg/1	SO <sub>4</sub> <sup>=</sup> mg/1	Cl <sup>-</sup> mg/1	NO <sub>3</sub> <sup>-</sup> mg/1	Na <sup>+</sup> mg/1	K <sup>+</sup> mg/1	Ca <sup>++</sup> mg/1	Mg <sup>++</sup> mg/1
1	163.64	-	16.60	6.71	33.39	16.33	2.08	39.96	18.9
2	43.36	-	654.58	3.67	10.15	24.35	8.77	148.27	74.6
3	87.62	-	187.04	6.50	6.21	11.34	7.91	58.29	26.7
4	96.62	-	9.93	1.54	8.84	5.39	5.28	15.19	11.9
5	55.25	-	791.59	8.47	7.01	38.81	9.38	159.08	94.5
6	77.99	-	148.10	3.68	12.07	13.54	3.81	42.56	22.83
7	88.49	-	5.99	2.31	12.25	0.96	4.16	26.33	6.88
8	72.14	-	29.45	3.03	3.14	7.40	2.48	16.92	9.72
9	94.64	-	49.92	3.60	5.00	6.72	6.17	26.65	13.06
10	104.22	-	37.44	1.43	2.42	4.99	7.76	27.98	10.40
11	105.62	-	65.73	0.63	1.21	6.54	7.13	35.35	14.44
12	75.53	-	317.35	1.44	1.99	72.44	4.52	43.91	35.73
13	137.89	-	83.20	2.20	4.00	27.84	9.16	30.28	19.54
14	150.77	-	416.00	2.00	1.65	144.31	8.47	48.82	29.89
15	89.71	-	274.56	8.38	3.40	60.47	7.15	37.21	33.59
16	88.74	-	29.95	1.61	2.56	12.82	4.34	17.66	9.55
17	66.09	-	181.71	1.74	1.53	30.59	7.19	44.89	20.66
18	84.58	-	24.96	1.00	1.90	9.19	4.46	16.71	9.11
19	71.51	-	362.75	0.46	0.60	4.88	7.12	89.68	43.99
20	82.06	-	85.86	0.29	3.28	8.40	13.23	26.90	14.80
21†	161.79	-	180.96	3.13	3.27	34.03	17.53	51.55	26.55
22	125.99	-	134.78	3.29	4.09	53.53	16.28	24.13	21.10
23	47.18	-	673.92	5.95	4.61	104.47	17.26	74.95	101.11
24	433.65	-	332.80	6.94	6.18	120.54	20.05	85.92	68.05
25	96.23	-	479.23	1.21	1.34	62.56	16.85	81.29	58.09
26	105.97	-	1,431.04	21.28	16.44	233.30	34.16	130.78	178.08
27	154.40	-	126.46	1.85	1.68	31.46	15.74	41.17	21.15
28	82.64	-	252.72	2.13	2.34	67.19	13.33	44.45	29.90
29	111.96	-	15.48	1.05	2.20	7.65	10.48	23.95	10.01
30	172.01	-	126.46	1.87	2.84	31.95	17.12	48.14	21.63
31	162.24	-	227.14	1.46	2.12	75.69	10.95	48.16	24.63
32†	57.58	15.13	42.93	3.01	3.85	41.86	12.47	5.94	4.94
33†	127.21	24.74	119.81	1.17	2.11	121.36	13.95	4.84	3.18
34†	252.69	21.18	74.88	19.23	9.89	178.24	6.61	6.49	1.68
35†	66.59	70.72	25.96	5.95	4.08	109.87	1.79	5.41	1.64
36†	183.77	15.94	164.02	0.32	1.87	151.14	1.76	-	2.45

† samples were subjected to static leaching

### 3.7.4 Sanitary and Other Wastes

Sanitary wastes and mine shop and maintenance area effluents previously were discussed in detail. Because no surface discharge into Girard Creek will be permitted, the sanitary wastes totalling about 11,000 m<sup>3</sup> (9.4 ac-ft) annually will probably be used in a spray irrigation system. Most of such a spray is evaporated, therefore only a very small input to groundwater is likely to result. No other waste products from the mine shop or maintenance area are expected to alter groundwater quality or quantity.

### 3.8 Deposition from Mining Emissions to the Atmosphere

In order to determine what effects air emissions from the mine may have on water quality in the Poplar River Power Project area, it is necessary to estimate both emissions and deposition of pollutants. The principal pollutant of interest with respect to this source is particulates. However, carbon monoxide (CO), sulfur oxides (SO<sub>X</sub>), nitrous oxides (NO<sub>X</sub>), and hydrocarbons (HC) also are of interest.

In regard to particulate emissions from the Coronach mine area, emission estimates were made for various mining-related activities (see Table 3-13). Emissions also were projected for vehicular traffic in the mine area and are shown in Table 3-14. The latter emissions are considered insignificant in terms of deposition rates and resultant water quality impacts; therefore, deposition estimates are provided only for particulates from the mining-related activities and are summarized in Table 3-15.

TABLE 3-13. Particulate Emissions from Mining Related Activities

ACTIVITY	EMISSION FACTOR	MULTIPLIER (tonnes/yr)	EMISSIONS (tons/yr)
DRAGLINES	0.012 kg/m <sup>3</sup> (0.021 lb/yd.) overburden removed	35.55x10 <sup>6</sup> m <sup>3</sup> (43.88x10 <sup>6</sup> yd.)	418
HAUL ROADS	3.16 kg (11.2 lb) per vehicle km (mile) traveled	1,560 km (970 miles)/shift	1,232
SHOVEL/TRUCK LOADING	0.001 kg/tonne (0.002 lb/ton) of lignite mined	3.4x10 <sup>6</sup> tonnes (3.8x10 <sup>6</sup> tons) mined per year	3.4
EXPOSED AREA	U. S. Dept. of Agriculture formula	129 ha (320 acres) 259 ha (640 acres) 405 ha (1,000 acres)	1,216 2,432 4,320
TRUCK DUMP	0.014 kg/tonne (0.027 lb/ton) lignite mined	3.4x10 <sup>6</sup> tonnes (3.8x10 <sup>6</sup> tons) mined per year	47
CRUSHER	0.1 kg/tonne (0.2 lb/ton) lignite crushed	3.4x10 <sup>6</sup> tonnes (3.8x10 <sup>6</sup> tons) mined per year	345
TRAIN LOADING	0.0001 kg/tonne (0.0002 lb per ton) lignite loaded	3.4x10 <sup>6</sup> tonnes (3.8x10 <sup>6</sup> tons) loaded per year	0.34
COAL STORAGE PILE AT LOADOUT FACILITY	1.8 v kg/hr/ha (1.6 v lb/hr/ acre	v is average wind velocity, 6 m/s	91.7
TOTAL EMISSIONS			
	for 129 ha (320 acres) exposed	3,232	3,561
	for 259 ha (640 acres) exposed	4,335	4,777
	for 405 ha (1,000 acres) exposed	6,048	6,665

TABLE 3-14. Emissions from Mine Vehicular Traffic

POLLUTANT	HEAVY DUTY TRUCKS (tonnes/yr)	HEAVY DUTY TRUCKS (tons/yr)	LIGHT WEIGHT TRUCKS (tonnes/yr)	LIGHT WEIGHT TRUCKS (tons/yr)	TOTAL EMISSIONS (tonnes/yr)
Carbon Monoxide (CO)	5.73	6.32	4.67	5.15	10.40
Nitrous Oxides ( $\text{NO}_x$ )	4.2	4.6	0.63	0.69	4.83
Hydrocarbons (HC)	0.9	1.0	0.14	0.15	1.04
Sulfur Oxides ( $\text{SO}_x$ )	0.56	0.62	0.08	0.09	0.64
Total Suspended Particulates (TSP)	0.25	0.28	0.04	0.04	0.29
					0.32

TABLE 3-15. Particulate Deposition Rates Associated with Mining Activity

DISTANCE DOWNWIND FROM MINING ACTIVITY (km)	DEPOSITION RATE (kg/ha/yr)
129 ha EXPOSED (320 acres)	259 ha EXPOSED (640 acres)
	405 ha EXPOSED (1,000 acres)
2	6.7
4	2.2
6	1.2
8	0.8
10	0.6
12	0.4
14	0.4
	12.7
	23.3
	4.4
	8.1
	4.4
	2.8
	2.0
	0.9
	1.6
	1.2
	0.7



## 4. TRANSPORTATION SYSTEMS

### 4.1 Description

Transport systems at the Coronach mine site and the Poplar River Power Station will consist of haulage from the mine to the loadout facility by truck, then transport of the lignite to the power station by unit coal train.

The only discharges considered from the transportation system were those from the road and railway corridor between the mine loadout and the plant loadout. It was assumed that all haul and access roads in the mine area proper will drain to the pit and therefore these will not be considered further here.

The road and railway between loadouts run parallel for about 8,000 m. The road surface is approximately 8 m wide, and the railway surface is about 7 m wide. Ditches located on each side of the road and railway occupy an area of approximately 10 m and carry runoff water back to natural drainage courses. Culverts are located at the crossing of each natural drainage basin and the road ditches would feed into these points. Alteration of normal drainage patterns should therefore be minimal. It is assumed that erosion control measures will be employed, ditches will be grassed and road dust will be controlled primarily with water.

### 4.2 Inputs and Losses to Surface Water Systems

In calculating the quantities of runoff water, the following assumptions have been made:

- (1) road surface runoff, 100%
- (2) railroad surface runoff, 5%
- (3) ditch area runoff, 50%
- (4) runoff from natural undeveloped land is 5.6% of the average total precipitation per year or 0.21 m.
- (5) runoff from the mine loadout loop and mine maintenance area would be segregated from the area being considered here.

Using this information, the quantities of surface runoff and net gain to the surface water system were calculated.

	<u>Transportation Routes</u>	<u>Original Developed Land</u>	<u>Net Gain/Year</u>
Average yearly precipitation	39,800 m <sup>3</sup>	4,200 m <sup>3</sup>	35,600 m

It should be noted that the increase shown is a result of higher runoff of precipitation from the road and ditches and that a portion of this runoff will be lost en route to Girard Creek. No significant change is predicted in the quality of runoff after the construction period. The transportation system effluents, therefore, should contribute only an increased quantity of water to Girard Creek, of a quality similar to natural runoff.

#### 4.3 Inputs and Losses to Groundwater System

The road surface and railroad right-of-way between the mine and the power plant will intercept some of the precipitation that might otherwise constitute an input to the groundwater system. This represents a loss to groundwater, but the quantity probably is very small and no attempt has been made to quantify this loss.

#### 4.4 Transportation System Emissions

Gross emissions for total suspended particulates, carbon monoxide, sulfur oxides, nitrous oxides, and hydrocarbons have been estimated for the two trains to be utilized for hauling lignite from the mine to the power plant (Table 4-1). No deposition estimates have been developed for these emissions since deposition rates and any resultant water quality impacts are not considered significant.

Table 4-1. Coal Train Emissions  
(Diesel emissions only, no lignite loss is assumed)

<u>POLLUTANT</u>	<u>EMISSIONS (tonnes/year)</u>
Carbon Monoxide (CO)	4.67
Total Suspended Particulates (TSP)	0.09
Sulfur Oxides (SO <sub>x</sub> )	0.08
Nitrous Oxides (NO <sub>x</sub> )	0.63
Hydrocarbons (HC)	0.14

5.1 Description5.1.1 General

A general layout of the major features of the Poplar River generating station is provided in Figure 5-1 and a view of the partially completed powerhouse is shown in Photo 5-1. Specific details of the immediate plant site are shown in Figure 5-2. The major features outlined in the figures are:

- (i) powerhouse (including service bay)
- (ii) stack and electrostatic precipitator
- (iii) switchyard
- (iv) water treatment plant
- (v) coal storage and handling area
- (vi) oil tank storage area
- (vii) cooling system (including intake, pumphouse, discharge channel and outlet structure)
- (viii) construction camp
- (ix) holding pond
- (x) evaporation pond
- (xi) sewage lagoon
- (xii) sod farm

Items (i) to (viii) are basic power station components whereas items (ix) to (xii) are waste disposal or containment facilities.

The remainder of Section 5.1 contains detailed descriptions of the major power plant features and the planned waste management facilities. Particular attention is paid to those aspects which could have a bearing on the water quality in Cookson Reservoir and/or the East Poplar River. Some of the features described are sub-components of the major features listed above. Estimation of the quantities, characteristics and destinations of plant effluents follow in Sections 5.2 and 5.3.

5.1.2 Condenser Cooling Water System(a) General

Condenser cooling water systems are used at thermal power plants to condense spent steam from the turbines back to water so that it can be recirculated through the steam cycle. Approximately 50 percent of the heat produced in coal combustion is disposed of in the cooling water discharge. The disposal of such large quantities of heat requires large cooling water volumes and a sufficiently large receiving water system to allow effective dissipation of the heat and heat transfer to the atmosphere. At fossil-fueled power stations, cooling water flow rates average about 28 l/s (1 cfs) per megawatt of power production.

PHOTO 5-1 POPLAR RIVER G.S. - PARTIALLY CONSTRUCTED POWERHOUSE IN MAY, 1978

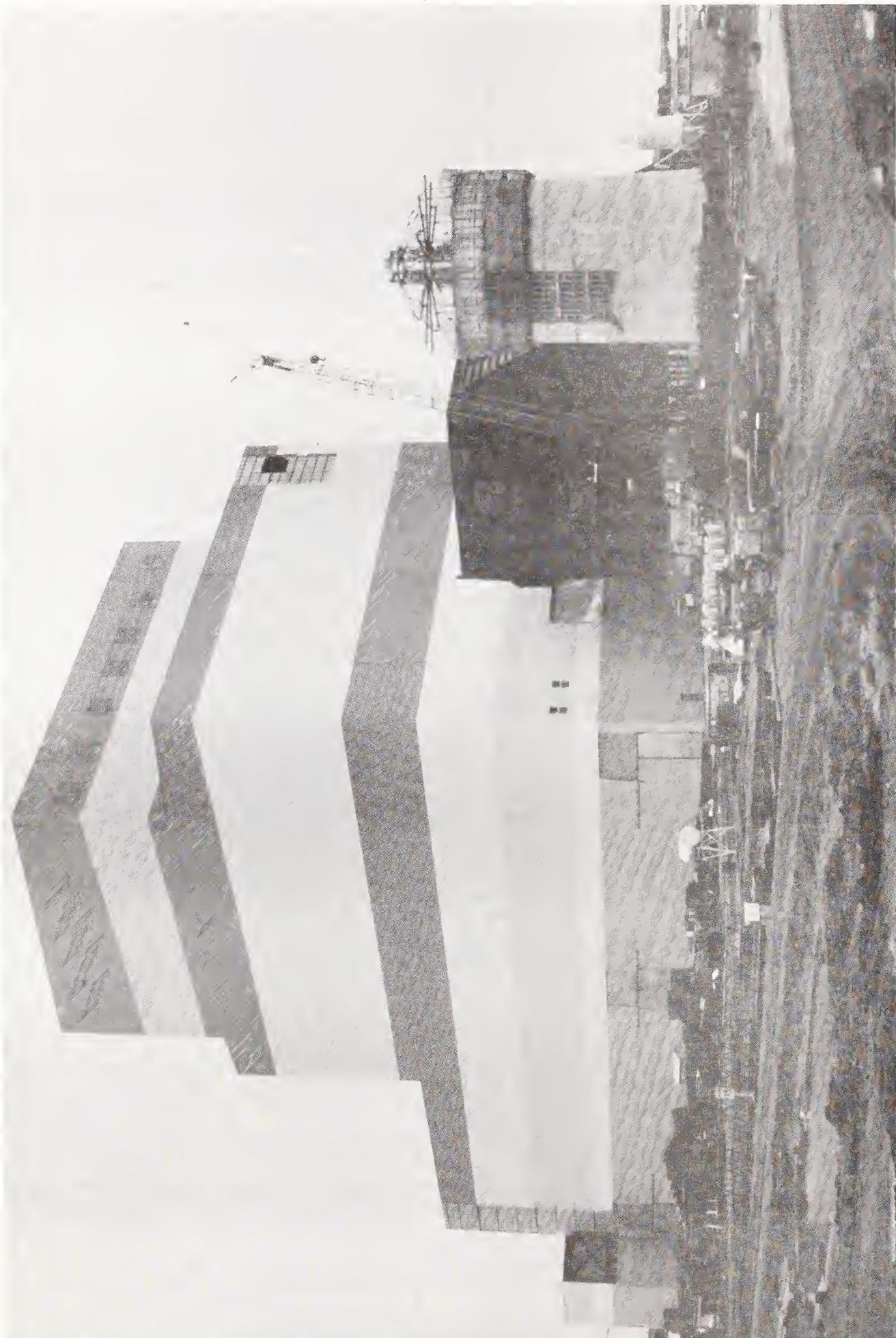
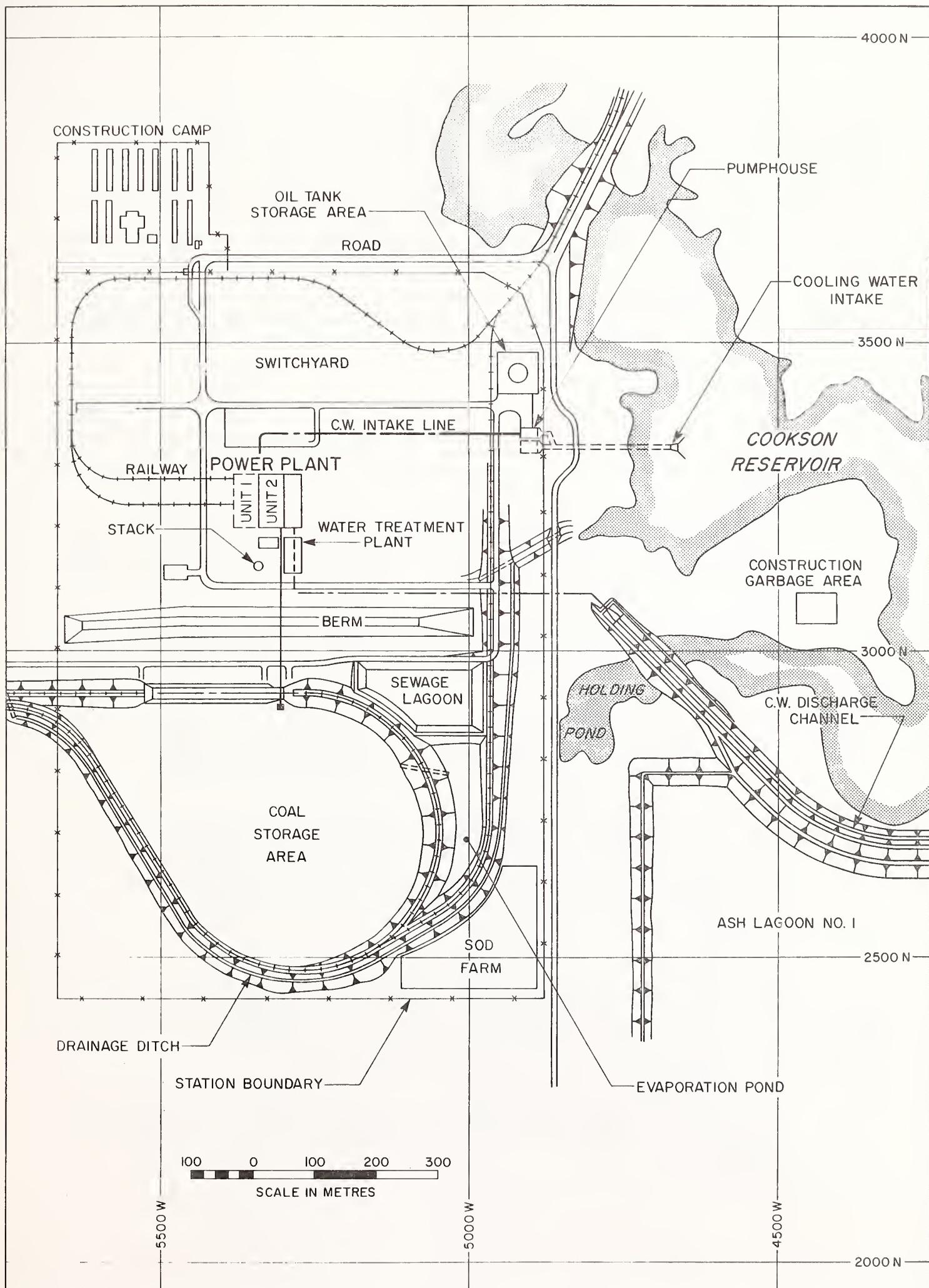


FIGURE 5-1 LOCATION OF MAJOR POWER PLANT FEATURES





FIGURE 5-2 LAYOUT OF POPLAR RIVER GENERATING STATION



Specific environmental concerns are normally identified with each of the major components of the condenser cooling system. Mortality of fish from impingement on the intake screens, from mechanical damage, chemicals or heat in passage through the cooling system, from exposure to elevated temperatures in the receiving water thermal plume or from cold shock resulting from power plant shutdown are among these concerns. Environmentally sound design of the cooling system will help minimize the adverse effects on aquatic populations.

The various components and method of operation of the cooling system for the Poplar River generating station are described in sections (b) through (e) below.

(b) Intake

The locations of the cooling water intake structure and intake pipe are shown in plan in Figure 5-3 and plan and profile views are presented in Figure 5-4. Pertinent dimensions and features of the intake are:

- (i) The 6.2 m diameter main intake pipe extends 186 m from the intake structure to a surge chamber in front of the location that would be occupied by the pumphouse for units three and four should they be constructed.
- (ii) The top of the intake pipe at the intake structure is at elevation 745.5 m, 0.5 m above the reservoir minimum pool elevation of 745.0 m
- (iii) The x-sectional area of the intake pipe is  $30.2 \text{ m}^2$ . Flow velocity in the pipe will be 0.51 mps when flow is equal to the design condenser flow rate for two units of 15,270 l/s.
- (iv) The elevation of the natural land surface leading into the intake along the bottom of Girard Creek valley is approximately 745 m, the reservoir minimum pool. This means that to operate at minimum pool, water would have to be conveyed to the intake via the normally submerged original Girard Creek channel.

(c) Pumphouse

The pumphouse location is shown in Figure 5-3, with plan and profile views shown in Figure 5-5. At ultimate development, one pumphouse would service units one and two and a separate adjacent pumphouse would service units three and four. Features to note are:

- (i) A 4.9 m diameter intake pipe for the pumphouse for units one and two branches from the main 6.2 m diameter intake pipe before it terminates in front of the location for the units three and four pumphouse

FIGURE 5-3 LOCATION OF COOLING WATER INTAKE AND OIL STORAGE AREA

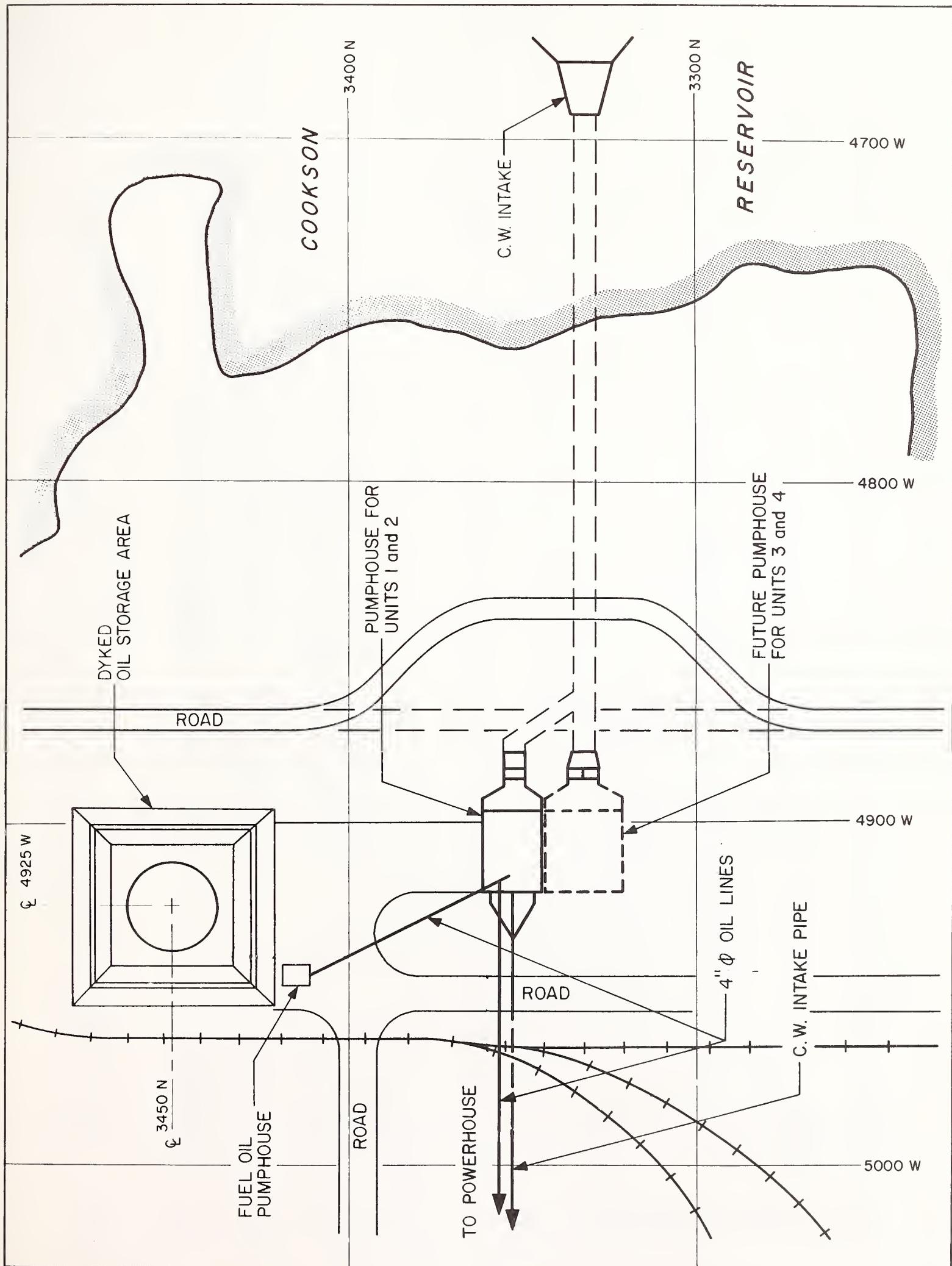


FIGURE 5-4 PLAN AND PROFILE OF COOLING WATER INTAKE

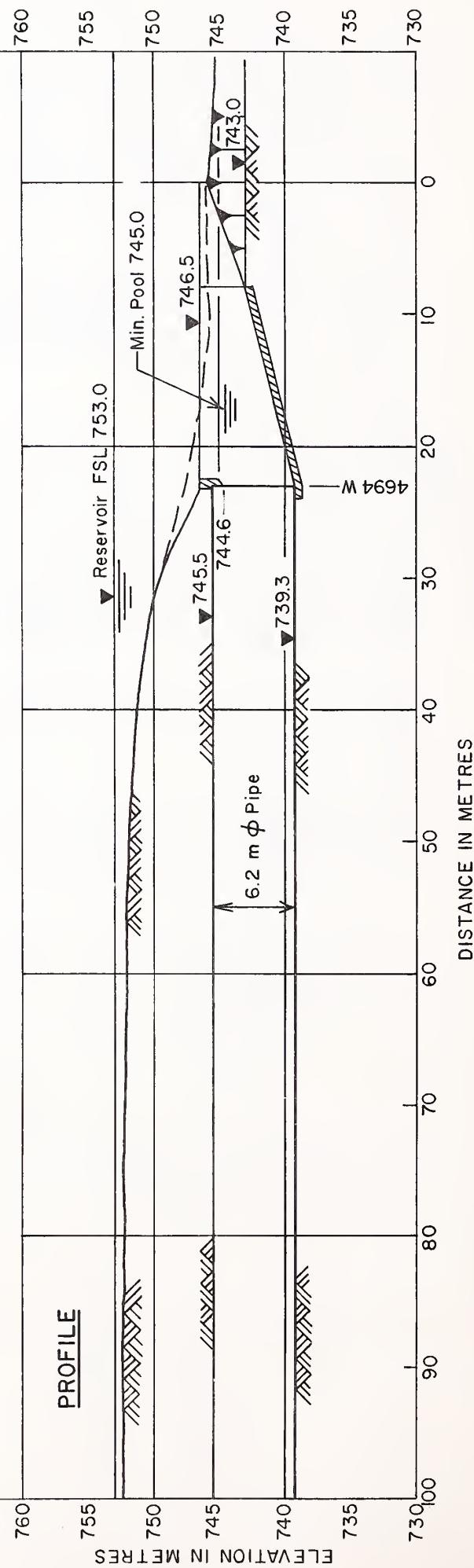
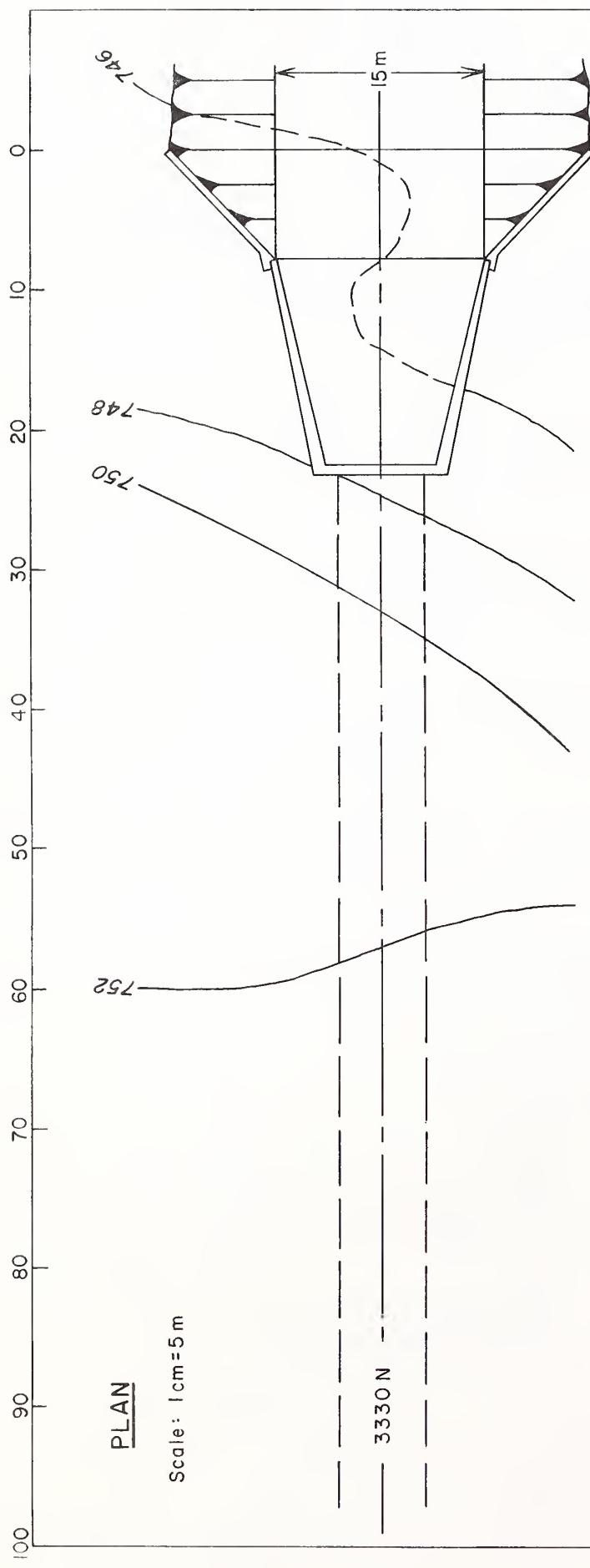
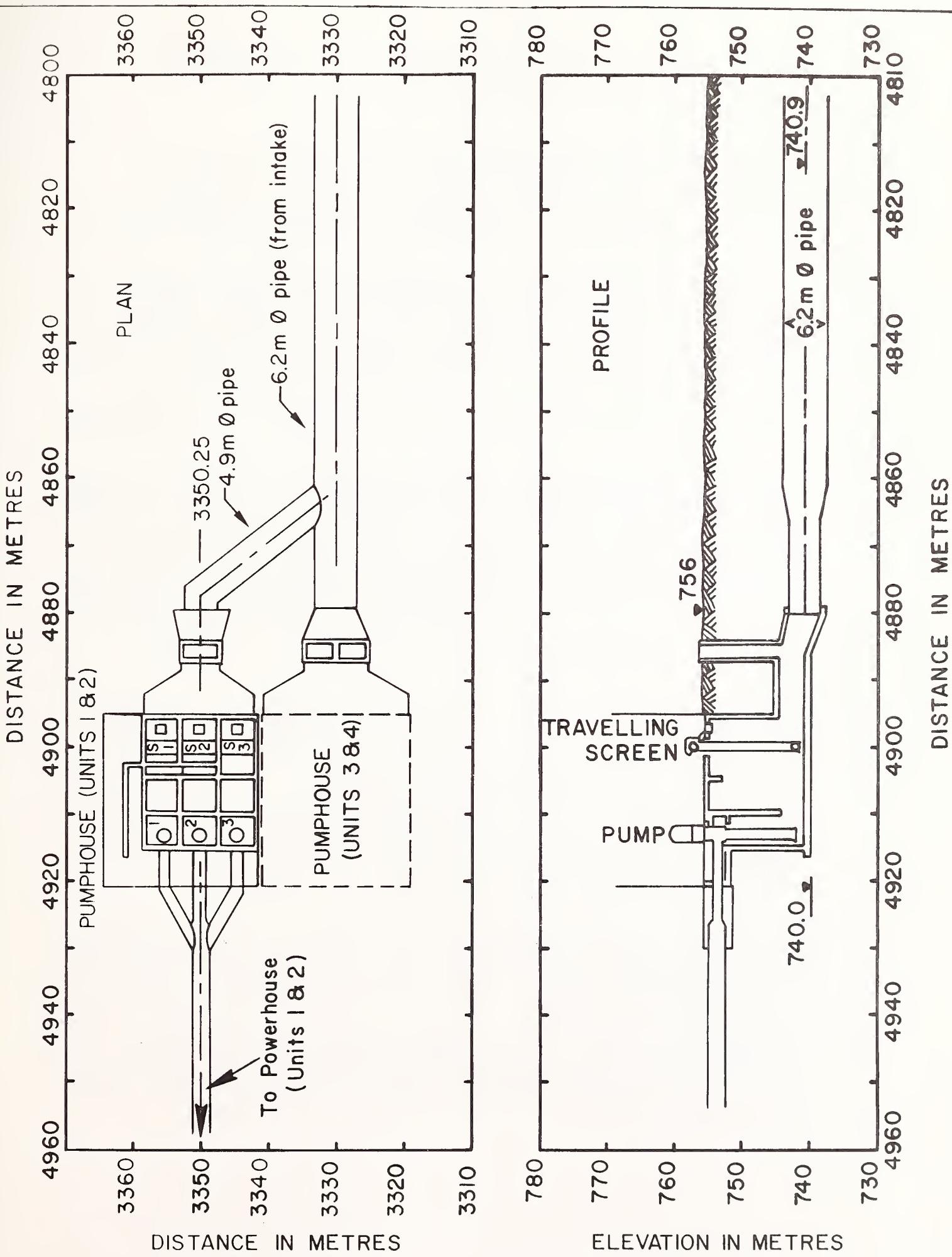


FIGURE 5-5 PLAN AND PROFILE OF COOLING WATER PUMPHOUSE



- (ii) The 4.9 m diameter pipe has a x-sectional area of  $18.8 \text{ m}^2$ . Flow velocity in the pipe would be 0.41 mps with one unit operating and 0.81 mps with two units operating.
- (iii) The pumphouse for units one and two contains three circulating water pumps. Each pump has a capacity of 6700 l/s with the reservoir at FSL elevation 753 m, 5500 l/s at design head elevation 748.3 and 400 l/s with the reservoir at minimum pool elevation 745 m as shown by the performance curves in Annex H. One pump will service one unit, except during periods of high reservoir water temperature. When the cooling water intake temperature is high, the standby pump (second pump with one unit, third pump with two units) will be brought into service. Actual cooling water flow will be throttled by a motorized valve located on the discharge end of the condenser and operated from the control room. The flow will be varied depending on the condenser vacuum, condenser flow limitations and intake water temperature.
- (iv) Each of the cooling water pumps is preceded by a 9.5 mm wire mesh vertical travelling screen. The screens are 4.1 m wide and the travelling screens span 14.94 m vertically. All three screens will be in use at all times. With three pumps operating, flow throttled to the condenser design rate for two units (15,270 l/s), 900 l/s added for auxiliary cooling water flow and uniform velocity distribution across the screens, the approach velocity to the screen with different reservoir levels are as follows:

Reservoir Level (m)	Flow (l/s)	Approach Area ( $\text{m}^2$ )	Approach Velocity (m/s)
753.0 (FSL)	16,170	147.6	0.11
750.0	16,170	110.7	0.15
748.3 (Pump Design Head)	16,170	89.8	0.19
745.0 (Minimum Pool)	12,000	49.2	0.35

Non-uniform velocity distribution when fewer than three pumps are in use could give rise to higher localized approach velocities for one of the screens.

Rotation of the travelling screens will be based on head loss through the screens caused by fish and debris impingement. Debris will be washed from the screens by water spray into trash bins located in front of each screen.

(c)

### Condenser

#### Design

A double pass condenser is proposed with each nest containing 6,793 tubes 9.144 m long and 25.4 mm in diameter. At a design flow of 7,635 l/s for each unit and an inlet temperature of 21.7°C, a temperature rise of 12.39°C will occur through the condenser. Using a design velocity of 2.22 m/s in the tubes (computed from the above tube diameters and flow rates), and assuming that the velocity should not exceed the maximum of 2.49 m/s listed in the condenser specifications, the maximum flow possible through each condenser would be 8,564 l/s. Since the capacity of each pump is only 6.7 l/s when the reservoir is at elevation 753 m, neither the design or maximum condenser flow will be reached without bringing the standby pump into service.

#### Anti-fouling

Chlorine is commonly used as a biocide to minimize fouling of the condenser tubes, tube sheets and water boxes by bacterial slime or other growths. The biological slime, in turn, tends to entrap particulate matter which can eventually build up to a dense scale like covering. The slime and scale deposits form an insulating layer which inhibits heat exchange in the condenser. The warmer water temperatures in the condenser promote more luxuriant growths of biological life.

The poorer the quality of the cooling water, the greater the chlorine demand. The food supply in poor quality water accelerates the growth of organisms in the condenser tubes increasing the required duration and/or frequency of chlorination to maintain satisfactory control. Bacterial growths can usually be effectively removed with a dosage sufficient to maintain a free available chlorine level of about 0.5 mg/l at the condenser outlet. The free available chlorine depends on the amount of chlorine demand agents and ammonia in the water. Ammonia reacts with chlorine and reduces the amount of free available chlorine by increasing the combined residual portion of total residual chlorine in the discharge.

Although the need for chlorination at the Poplar River station is not clearly established, expected reservoir nutrient levels are likely to require some chlorine dosage. The application procedure suggested calls for a controlled injection of 5 mg/l for three 10 minute periods per day for each unit. The chlorine injection point will be just before the condenser and hence minimize dosage requirements to maintain an acceptable chlorine residual. With two units operating, residual chlorine in the cooling water discharge will be further reduced by treating only one unit at a time with dilution by discharge from the unit not being treated.

### Scale Prevention

Condenser tube scaling is caused by calcium carbonate deposition on the tube walls. It is usually prevented by controlling the hardness and alkalinity of the cooling water and a variety of methods are available to do this. Since scaling may be a problem at the Poplar River station, scale control options are being considered but no firm decision has been taken on the most suitable option. One alternative being considered is softening of the reservoir water by a cold lime treatment process. With this process it may be necessary as well to add sulphuric acid to effect the necessary reduction in alkalinity during an extended dry period. A softening facility would be required that would draw a portion of the cooling water flow from the discharge side of the condenser, treat it and release it back into the discharge channel. Sludge from the treatment plant would be run into the ash lagoons along with the ash slurry or disposed of in a separate lagoon, the former being most probable. Sludge quantities would vary depending on reservoir water quality and can best be estimated by running the reservoir operation model. The process essentially involves softening of the entire reservoir water volume and is dealt with further in Section 6 on Reservoir Operations and Section 8 on Mitigation Measures.

Other methods of scale control are possible including the use of scale inhibitory chemicals and mechanical cleaning methods. Polyelectrolytes are often used as dispersing agents to retard the precipitation of calcium carbonate. These typically include individual or combined compounds of zinc, chromate, hexametaphosphate, phosphonate and polyesters (EPA, 1974), some of which could lead to undesirable concentrations of certain elements in the reservoir. There is a possibility that the polyelectrolyte being used at the Boundary Dam station, Al Chem 5WP549, an ammonium chloride compound, will be used at Poplar River at an annual rate of 91,000 kg per year for two units. The U.S. Environmental Protection Agency has approved its use in potable water at concentrations up to 50 ppm.

It may be possible, as well, to use minimum chemical treatment by acid addition to control the stability index at a slightly scaling condition. Mild descaling could then be accomplished with occasional pH depression to about 5.0 or the use of on-line condenser tube cleaning equipment such as Amertap. The sponge balls of the Amertap system are effective in removing soft chemical precipitates or bacterial slimes before they become adherent and are furnished with an abrasive band to remove older deposits by a scouring action. Minimum on-line chemical cleaning and on-line mechanical cleaning could be supplemented by either chemical or mechanical cleaning during scheduled outages, although the manufacturer claims that Amertap often eliminates the need for manual cleaning.

Final selection of a method of scale control at Poplar River may not be made until operating experience has been obtained with the first unit.

(e) Discharge

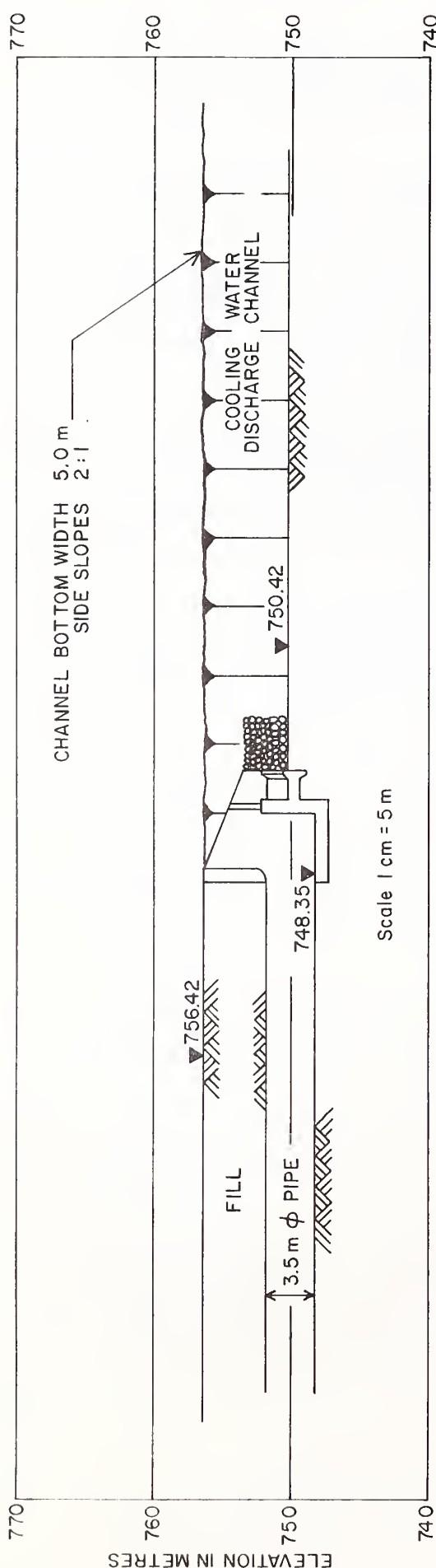
The major facilities of the cooling water discharge system are:

- (i) A 3.05 m diameter pipe from the powerhouse to the upper end of the discharge channel
- (ii) An outlet structure for the transition from the pipe to the discharge channel (Figures 5-1 and 5-6, Photo 5-2)
- (iii) A channel 3,800 m long with a 5 m wide base and 2:1 sideslopes leading from the pipe-to-channel outlet in a southeasterly direction along the strip of land between the reservoir and the ash lagoons (Figure 5-1)
- (iv) A 3 m x 5 m box outlet structure at the lower end of the discharge channel leading due east along 700 N into the reservoir (Figures 5-1, 5-6). Some of the pertinent features of these facilities are shown on the figures indicated. The estimated flow velocities are as follows:

	Design Flow 8,500 l/s <u>1 Unit</u>	Design Flow 17,000 l/s <u>2 Units</u>
3.05 m diameter pipe between plant and channel	1.13 m/s	2.26 m/s
Over discharge weir into channel		
1) with reservoir level 753.0	1.10 m/s	1.80 m/s
2) with reservoir level 745.0	--	6.89 m/s
Discharge Channel		
1) with reservoir level 753.0	0.24 m/s	0.49 m/s
2) with reservoir level 745.0	0.49 m/s	0.58 m/s
Discharge culvert into reservoir		
1) with reservoir level 753.0	0.58 m/s	1.13 m/s
2) with reservoir level 745.0	--	8.92 m/s

Through part of its length, the discharge channel will be an excavated section at the toe of the ash lagoon dike. Near the lower end, after it pulls away from the ash lagoons, the channel will be a cut and fill

FIGURE 5-6 COOLING WATER DISCHARGE OUTLET STRUCTURES



OUTLET STRUCTURE - DISCHARGE PIPE TO CHANNEL

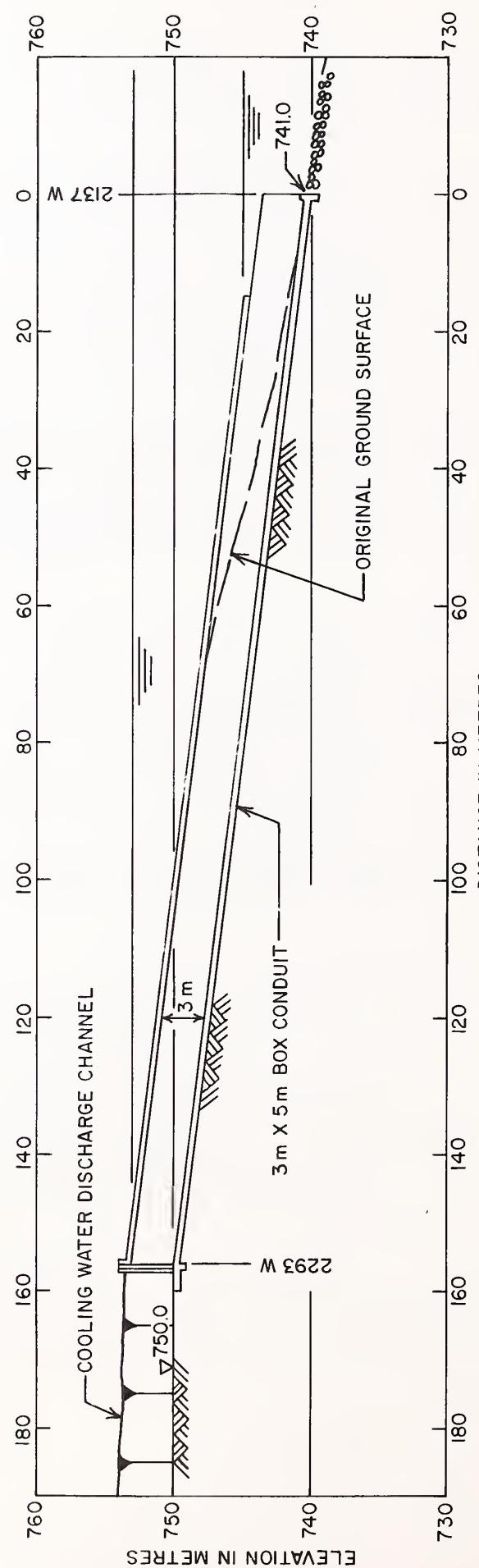


PHOTO 5-2 OUTLET OF COOLING WATER DISCHARGE PIPE TO COOLING  
WATER DISCHARGE CHANNEL DURING CONSTRUCTION IN MAY, 1978



section with a low dike on the west side. At the upper end, the bed of a creek which originally flowed eastward through the power station site area and into Girard Creek crosses the channel route. Across the creek bed the channel will be totally contained by parallel dikes, which will be flanked on the west side by the holding pond and on the east side by the reservoir.

#### 5.1.3      Auxiliary Cooling Water System

Auxiliary cooling water is water used for cooling various types of equipment in the plant such as hoppers, bearings and pumps. At Poplar River, auxiliary cooling water is drawn from the condenser cooling water inlet line at a rate of approximately 450 l/s per unit and run through an open loop system (Saskmont, Oct., 1978). Except for a small quantity drawn for the water treatment plant, the remainder of the open loop flow is used to cool water in a recirculating (closed loop) cooling system, which is used for all oil cooling purposes. The plate type water-to-water heat exchangers used to transfer heat from the recirculating to the open loop system are designed to prevent water leaks between systems. Any gasket or joint leaks drain onto the floor and therefore the possibility of the closed loop system contaminating the open loop system is remote.

Any oil leak into the closed loop system will increase the level in the head tank and initiate an alarm. Should it be necessary to drain the closed loop circuit, because of an oil leak or any other reason, this can be accomplished by draining to the main boiler house sump, where any oil contamination will be removed before discharge to the ash lagoon.

Before being discharged, a portion of the 450 l/s per unit auxiliary cooling water flow is diverted for other purposes. Approximately 130 l/s is diverted to the ash sluice system, 30 l/s is taken for ash disposal system cooling, 5.3 l/s goes to the water treatment plant and 4.7 l/s is consumed in other uses such as floor drains, laboratory wastes, condensate polishing plant, sumps, coal gallery drains, boiler fireside washing and air preheater washing. This leaves about 280 l/s for direct discharge into the main cooling water return.

The 30 l/s diverted to the ash disposal cooling system is used for furnace cooling, boiler ash hopper sealing, ash pump bearing cooling and economizer ash tank cooling. After use, the ash system cooling water is discharged to the main boiler house sump and subsequently to the ash lagoon along with other wastes contained in the sump. A very small portion, approximately 0.6 l/s average per unit, of the ash system cooling water is diverted to the economizer grits holding tank to make up for evaporative losses to the flue gas. It is expected that most of the water supplied to the economizer tank will be evaporated with only emergency overflow ending up in the main boiler house sump.

#### 5.1.4 Water Treatment Plant

The water treatment plant is made up of three parts:

- (i) a pretreatment plant consisting of a clarifier, gravity filters, carbon filters and a softwell.
- (ii) reverse osmosis equipment containing high pressure pumps, cartridge filters, reverse osmosis units, a holding tank and reverse osmosis cleaning equipment.
- (iii) a demineralizing plant containing cation units, decarbonator, anion units and mixed bed units.

The reverse osmosis unit is being used because dissolved solids concentration levels in the reservoir are expected to be too high for economic removal by normal ion exchange methods (SPC, Jan., 1978). Before water enters the reverse osmosis units, it is treated with sulphuric acid to prevent fouling of the equipment. The quantity of acid required is determined by the calcium and alkalinity concentrations. The reverse osmosis unit will reduce dissolved solids levels by more than 90 percent before the water is demineralized for use as boiler feedwater makeup.

The demineralizing plant is used for final preparation of boiler feedwater as well as recycling of a portion of the boiler condensate for polishing in the mixed bed units. Sulphuric acid and sodium hydroxide are used to regenerate both the demineralizer units and the condensate polishing units.

The chemicals that will be used annually in water treatment are listed below:

	<u>1 unit</u>	<u>2 units</u>
Clarifier - polyelectrolyte	9,070 kg	18,140 kg
- chlorine	1,360 kg	2,720 kg
Reverse osmosis - 39% sulphuric acid	13,610 kg	27,220 kg
- 50% caustic soda	29,480 kg	58,970 kg
Condensate polishing - 93% sulphuric acid	21,090 kg	42,180 kg
- 50% caustic soda	21,090 kg	42,180 kg
- 35% ammonia	45 kg	90 kg

Intake to the water treatment plant must be sufficient to balance off boiler water makeup requirements and water lost in wastes discharged from the plant. The total intake is estimated at 10.3 l/s for 1 unit and 20.6 l/s for 2 units. To control water temperature a portion of these flows is taken from the auxiliary cooling water flow intake and the remainder from

the heated condenser discharge water. These proportions depend on ambient water temperature which varies throughout the year. It is assumed that 5 1/s for 1 unit and 10 1/s for 2 units is drawn on average from the main cooling water return.

Wastes generated in the water treatment plant consist of waste from the demineralizing plant, reverse osmosis rejects and discharge from the neutral and floor drains. The reverse osmosis rejects are sent directly to the cooling water discharge. Effluent from the demineralizing plant is sent to a neutralization sump for pH adjustment to the 5.5 to 9.5 range. pH meters are used to determine whether the wastewater has been successfully neutralized, in which case it will be transported to the ash lagoon. If not properly neutralized, it will be released to the holding pond for further treatment.

Large acid and caustic soda storage tanks will be housed in the water treatment plant. These will be positioned above the neutralization sump to ensure that any accidental release or spill can be contained.

The plant laboratory is located in the water treatment building. Analysis to be carried out in the laboratory are expected to include make-up water analysis, coal analysis, ash analysis, holding pond water analysis, and ash lagoon water analysis. The wastes generated in the laboratory will be collected in the neutral drains sump along with wastes from other plant drains. The neutral drains sump in turn discharges to the ash lagoon.

#### 5.1.5 Oil tank Storage and Oil Transport Lines

The oil tank storage area located northeast of the station is shown in Figure 5-3. The oil tank is contained within a 2300 m<sup>2</sup> area enclosed by a perimeter dike. Rainfall water, and any accidental releases or spills, are contained by the dike enclosed area which has sufficient capacity to contain the total contents of the tank. The earth dikes will be lined with till to prevent seepage of oil into the soil. No drainage system is provided and it will be necessary for the catchment area to be pumped out and the contents disposed of depending on the constituents.

Oil is pumped to the powerhouse and cooling water pumphouse from a fuel oil pumphouse located on the south side of the oil storage area. A 10.2 cm diameter pipe runs underground to the cooling water pumphouse where a 2.5 cm diameter pipe is tapped off. The 10.2 cm line continues to the powerhouse via the cable tunnel.

#### 5.1.6 Transformers

Indoor transformers are expected to be air cooled and outdoor transformers will be cooled with mineral oil. The foundation of each oil cooled transformer is sized to contain more than the total volume of oil contained in the transformer. Under normal operating conditions any water

entering the foundation will drain through holes provided and seep through sand fill surrounding the foundation. Any oil leaks will drain into the sand fill but are not expected to pass through the sand. Ultimately the sand will become plugged with oil necessitating that the sand be replaced. Any catastrophic event which fills the foundation will require special consideration being given to the disposal of the contents.

#### 5.1.7 Boiler

##### (a) Boiler Blowdown

The locations of the boiler and boiler blowdown tank are shown in Figures 5-7 and 5-8. Blowdown is expected to be intermittent, perhaps one day per week when the silica concentration reaches about 1 ppm. The blowdown water is cooled by a temperature controlled injection of auxiliary cooling water. Ammonia and hydrazine are added to the recirculating boiler water for pH adjustment and oxygen removal to prevent corrosion of metallic surfaces. The quantities added annually are as follows:

	<u>1 Unit</u>	<u>2 Units</u>
35% ammonia (NH <sub>4</sub> OH).....	1,815 kg	3,630 kg
30% hydrazine (N <sub>2</sub> H <sub>4</sub> ).....	2,270 kg	4,540 kg

Through a combination of makeup water treatment and condensate polishing, the solids level in the boiler water will be maintained near zero. Low level total phosphorus treatment will be used only under emergency conditions when condenser leakage is experienced. At temperatures above 260°C, some decomposition of the added hydrazine into ammonia can be expected making total ammonia in the boiler blowdown somewhat higher than the direct ammonia addition quantities listed above.

Because of reduced solubility at high operating temperatures, blowdown of boiler water is needed to maintain acceptable levels of scale forming salts. The boiler blowdown is expected to have a pH of 9 and be oil free. Blowdown rates are estimated at approximately 4.8 l/s and could average as low as 1 l/s. At 4.8 l/s, total ammonia in the blowdown would average about 2.6 mg/l. Blowdown water will be conveyed to the main boiler house sump via the blowdown tank.

##### (b) Boiler Waterside Cleaning

Preoperational boiler cleaning will be experienced only once in the life of the boiler. This typically consists of a series of chemical applications and rinses to remove mill scale, oil, grease and construction debris. Chemicals to be used in the preoperational cleaning at Poplar River are as follows:

FIGURE 5-7 PLAN VIEW - POWERHOUSE BASEMENT LEVEL

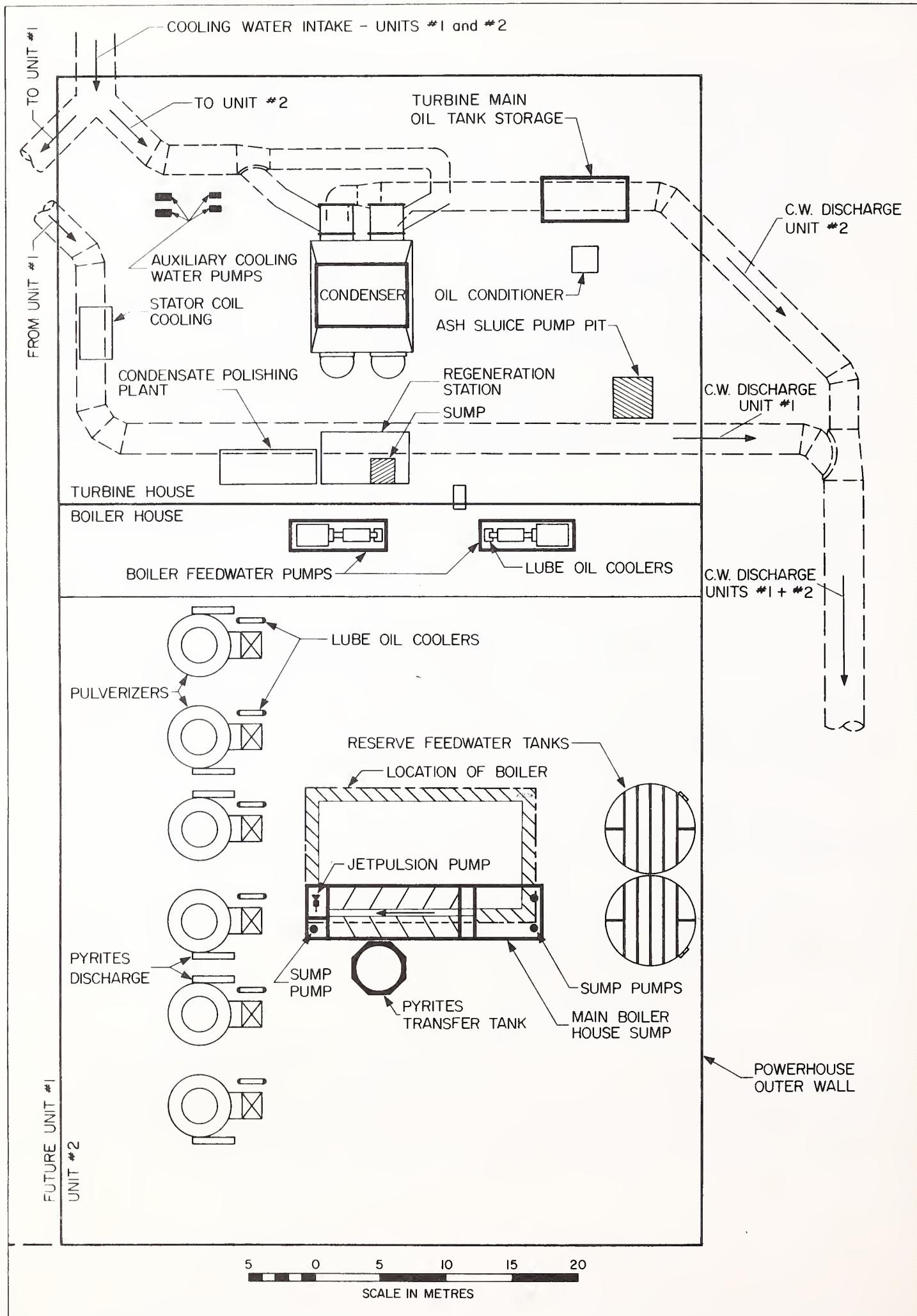
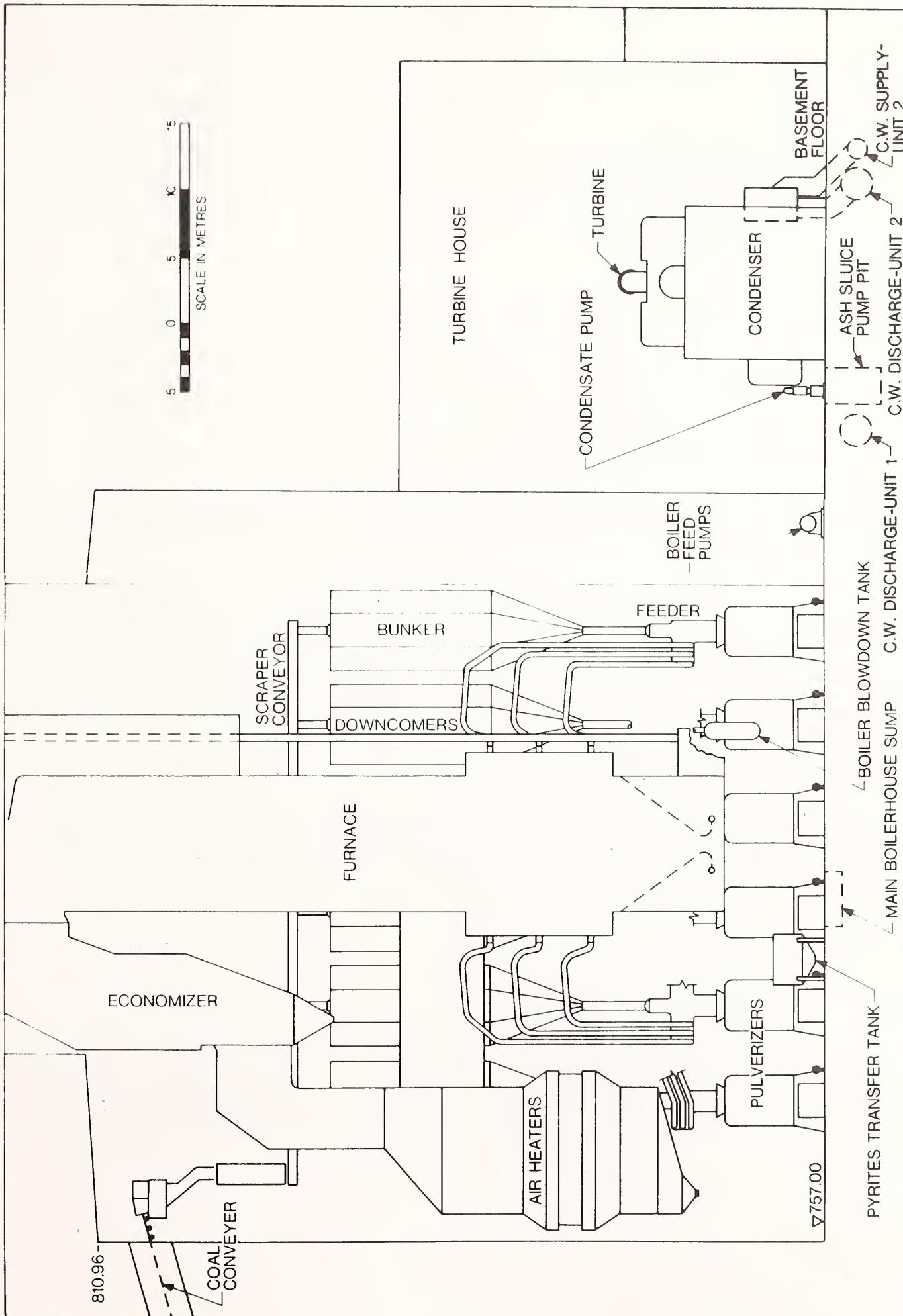


FIGURE 5-8 PROFILE OF POWERHOUSE UNIT NO. 2



	<u>1 unit</u>	<u>2 units</u>
sodium carbonate.....	2,166 kg	4,310 kg
trisodium phosphate.....	1,590 kg	3,180 kg
surfactant.....	90 l	180 l
citric acid.....	1,590 kg	3,180 kg
inhibitor.....	225 kg	450 kg

	<u>1 unit</u>	<u>2 units</u>
anhydrous ammonia.....	455 kg	910 kg
sodium nitrate.....	455 kg	910 kg
35% hydrochloric acid.....	7,940 kg	15,880 kg
ammonium bifluoride.....	385 kg	770 kg
35% hydrazine.....	23 kg	45 kg

The boiler preparation contract specifies that preoperational boiler cleaning wastes will be disposed of by the contractor at an offsite location.

Post operational boiler cleaning will be very infrequent. At the Boundary Dam generating station, for instance, this has been done only once in 18 years of operation. Citric acid-ammonia solutions will be used for such cleanings and this reagent will be disposed of by combustion in the boiler furnace. However, approximately 6,800 kg of chloride ion will be discharged after a boiler clean.

(c) Boiler Fireside Washing

Boiler fireside and air preheater washing will be carried out approximately once per year at 45 l/s (600 Igpm) for 48 hours with water being discharged to the bottom ash hopper. On an annual bases, the wash water totals 7,850 m<sup>3</sup> per unit with an average flow rate of 0.25 l/s per unit.

5.1.8 Powerhouse Drains and Sumps

(a) Wastewater Collection

Floor drains lead to strategically positioned sumps in the turbine house and boiler house. The wastewaters collected by floor drains typically contain dust, flyash, coal dust, floor scrubbing detergent, lubricating oil or other oils washed away during equipment cleaning, oil

from leakage of pump seals, oil from spillage around storage tank areas and other similar wastes. From the turbine house and boiler house sumps, and from sumps in the service bay and control annex, the collected wastewater is pumped to the main boiler house sump for treatment before release to the ash lagoons. Although pumps proposed for transferring wastewater from individual sumps to the main boiler house sump have capacities totalling 36 l/s per unit, the average release from all sumps is not expected to exceed 1 l/s per unit.

(b) Main Boiler House Sump

Inputs to the main boiler house sump other than those identified in (a) have all been mentioned previously. These include the ash system cooling water, boiler blowdown and economizer grits holding tank overflow. The largest of these by far is the ash system cooling water at about 30 l/s per unit followed by boiler blowdown at about 4.8 l/s.

Figure 5-7 shows the location of the main boiler house sump and several other containment facilities on a plan of the powerhouse basement level. Some of these features are shown as well in elevation in Figure 5-8. The overall dimensions of the sump are 18 m long x 4 m wide x 1.37 m deep. The sump is designed to hold 100 m<sup>3</sup> of water and is divided into two sections, a clean side and a dirty side, by a double weir. Water flows under and over the weirs from the dirty to the clean side while the oil and dust are retained in the dirty side (Saskmont, Oct., 1978).

The dirty side of the sump is treated through a filter and oily water separation system by a continuous circulation of the top layer of dirty water. The oily water separator takes out the oil and deposits it into a storage tank for disposal by road transport. The clean returns from the separator having less than 15 ppm of oil are returned to the dirty side of the sump for recycling. The clean water coming over the weir into the clean side of the sump is pumped down the discharge line to the ash lagoon. The capacity of these pumps is expected to be 180 l/s per unit consisting of one duty pump and one spare. The ash and sludge which collects in the bottom of the dirty sump is collected by the ash handling system and returned to the ash lagoons via the pyrites transfer tank.

(c) Pyrites Transfer Tank

Pyrites in general is rock in the coal that won't pulverize. Pyrites ejected from the pulverizers are collected in storage tanks attached to each pulverizer (6 per unit). The tanks have a 4 to 8 hour storage capacity. 6 times each day the pyrites are transported by ejector pumps to the pyrites transfer tank. Once each shift or 3 times per day the transfer tank is in turn emptied by an ejector pump into the ash slurry tank. It is then conveyed along with the ash to the ash lagoons via the ash slurry discharge pipe.

Overflow from the pyrites transfer tank is emptied into the dirty side of the main boiler house sump. This overflow should be relatively clean since the transfer tank acts as a decanter separating the large, heavy pyrites from the transport water.

#### 5.1.9     Outside Storm Drains and Roof Drains

The drains collecting storm runoff from the station site area discharge directly into a ditch which empties into the reservoir just northeast of the power plant. Roof drains discharge into the cooling water discharge pipe. Average annual flow rates expected from these drains are 0.6 l/s per unit from roof drains and 0.5 l/s for either 1 or 2 units from outside storm drains, based on an average annual precipitation rate of 0.355 m. The 15-minute maximum flow rate from roof drains is estimated to be 47.3 l/s per unit.

The Girard Creek tributary which originally flowed across the north part of the coal storage area is diverted around the area via a ditch along the south side of the perimeter rail grade. The ditch continues due north along the east side of the station site to enter an existing ditch leading in a northeasterly direction. The existing ditch, which collects other runoff from the site, flows through a culvert under the existing municipal road and into the reservoir.

#### 5.1.10    Sanitary and Solid Wastes

##### (a)       Construction Camp

The construction camp, located north of the station as shown in Figure 5-2, does not have any form of storm drainage system. Runoff flows eastward into the Girard Creek branch of Cookson Reservoir. This drainage from the construction camp area should not be significantly different from natural runoff from this area.

The camp's water supply is obtained from wells near the camp. At an average camp population of 300 and a per capita flow of 295 l/day (65 Igpd) total consumptive use of water is 1.03 l/s (19,500 Igpd).

All wastewater from the camp is directed to a temporary sewage lagoon located northeast of the camp site. There has not been any direct discharge from the lagoon since seepage into the ground is high. When the power plant's sewage treatment and effluent irrigation system is operational, the temporary lagoon will be filled in and abandoned, or may be partially used during construction of the second unit.

SPC has an agreement with the Village of Coronach to deposit all putrescible organic garbage in Coronach's municipal landfill on SW 18-2-26-W2 near Coronach. All inert construction material, wood, paper, etc., is being deposited and periodically burned in a pit near the inlet of the cooling water return canal (see Figure 5-2). The material should not significantly affect ground water or the water in Cookson Reservoir.

(b)

### Power Plant

The proposed sewage treatment and disposal system includes a packaged activated sludge treatment plant, an effluent holding lagoon and effluent irrigation (see "Engineering Report for Sewage Disposal" and "Sewage Treatment Plant Specifications" in Annex H and Figure 5-2). The sewage works are designed to handle the wastewater from the temporary camp and a power plant operation work force of 130. The maximum flow rate is estimated to be 1.85 l/s and the average annual rate 0.52 l/s.

The clay lined sewage lagoon is located between the coal storage area and the holding pond. A view of the empty lagoon during plant construction is shown in Photo 5-3. An approval to construct (under Saskatchewan Regulation 88/68) the collection system, pumping stations, treatment plant and a clay-lined holding pond has been issued by the Saskatchewan Department of the Environment.

Water from the sewage lagoon will be used to irrigate the sod farm located on the southeast corner of the station site. The 30,000 m<sup>2</sup> irrigation area shown is not considered adequate to handle the system design flow. It is also immediately adjacent to the excavated drainage ditch along the coal storage area perimeter grade, which drains directly into Cookson Reservoir. The subsurface conditions are not known since no soils or geological work have been done within the irrigation area. Drill hole RG1172, completed as part of the ash lagoon area geological investigation, is located just off the southeast corner of the irrigation area. RG1172 shows sand and silt in the upper till layer to be slight. However, other test holes in the coal handling area and farther east at the north end of the ash lagoons indicate that the till layer in that general area can be inconsistent with localized areas of substantial sand and gravel inclusion. This type of formation in the irrigation area cannot be ruled out without additional geological work.

As a result of these concerns, the SPC has been asked (i) to redesign the irrigation system and (ii) obtain and submit information regarding soil conditions, drainage and hydrogeology at the site in accordance with Environment Saskatchewan's "Sewage Effluent Irrigation Guidelines". When the information is received and it is determined that the soil is amenable to irrigation with sewage effluent and groundwater will not be significantly affected, an approval to construct will be issued.

Approval to operate both the lagoon and the irrigation system will be subsequently issued provided conditions are met regarding operation and monitoring.

#### 5.1.11 Holding Pond

(a)

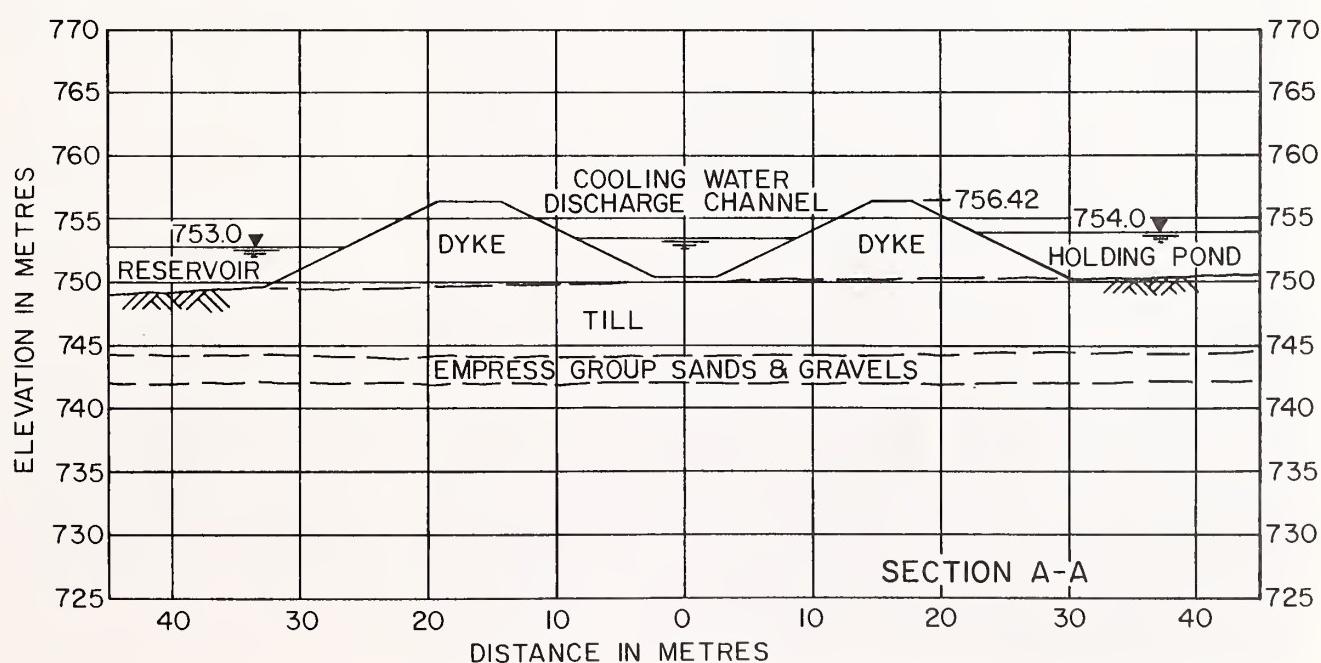
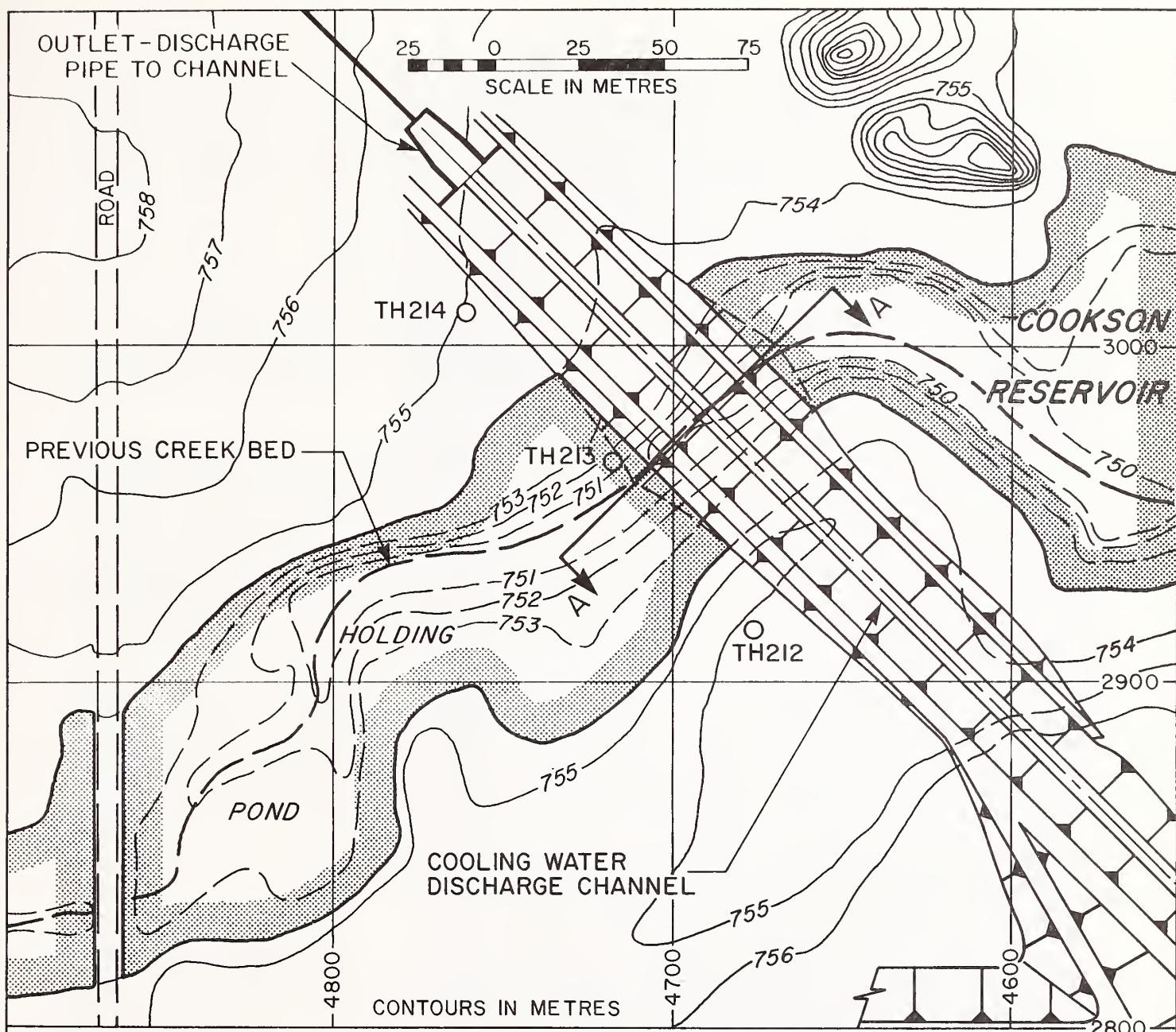
##### Location and Dimensions

The waste water holding pond is located just east of the coal storage area in the section of a former creek bed bounded by the road along the eastern boundary of the site and the dike containing the upper end of the cooling water discharge channel. The holding pond is shown in Figure 5-9

PHOTO 5-3 VIEW OF EMPTY SEWAGE LAGOON DURING POWER  
PLANT CONSTRUCTION IN MAY, 1978



FIGURE 5-9 HOLDING POND - PLAN AND PROFILE



and Photo 5-4 shows the pond filled with local runoff water during construction of the first power plant unit in the spring of 1978. It has a capacity of approximately 20,000 m<sup>3</sup> at its design full level of 754 m, and a maximum depth of 4 m.

(b) Subsurface Conditions

The locations of three test holes along the cooling water discharge channel west dike are also shown on Figure 5-9. One of these, TH213, is inside the holding pond area and the other two are just outside of the pond on either side.

TH213 shows approximately 2m of sand and gravel over underlying till. TH212 just south of the lagoon shows a similar formation with about 1.5 m of surficial sand and gravel while TH214 north of the lagoon indicates essentially till throughout. The maximum depth of each of the holes is approximately 6 m so none are deep enough to reach the sand and gravel layer (Empress Group) that typically underlays the upper till layer throughout the area.

It is expected that the surficial sands and gravels in the creek bed will be excavated for the cooling water discharge channel dyke foundations and the dykes placed directly on the till. This, in combination with the low head differentials between the holding pond, discharge channel and reservoir water levels, is expected to keep seepage loss from the pond low.

(c) Waste Inputs

Wastewater will be discharged to the holding pond only if some special contamination problem occurs with one of the normal waste releases to the ash lagoon. Water from the demineralization plant that has not been properly neutralized, unusual releases from the main boiler house sump or short term discharge of ash slurry water that has for some reason become contaminated are possible inputs to the holding pond. Wastewater discharge into the holding pond will be treated as required before release into the ash lagoon.

5.1.12 Coal Storage, Handling and Characteristics

(a) Reserve Coal Stockpile

The coal storage area within the circular rail grade is approximately 200,000 m<sup>2</sup>. The volume of coal stored will be 225,000 tonnes for 1 unit and 450,000 tonnes for 2 units, which represents a 30-day supply. The coal pile will have a raised center and slope down radially toward the base of the perimeter rail grade. Maximum coal size in the pile will be 15 cm. The pile will be compacted to make it fairly impervious. The environmental assessment report (SPC, Jan., 1978) suggests there is an aquifer under pressure beneath the coal pile, probably less than 5 m down.

PHOTO 5-4 HOLDING POND FILLED WITH LOCAL RUNOFF  
DURING PLANT CONSTRUCTION IN MAY, 1978



Runoff from the coal pile is expected to average about 35,000 m<sup>3</sup>/year or 1.1 l/s. It will be collected in drains and directed to the evaporation pond immediately east of the coal storage area (Figure 5-2). The evaporation pond shown in Photo 5-5 has a capacity of 25,000 m<sup>3</sup>. It may be necessary in years of high precipitation to pump wastewater from the evaporation pond to the ash lagoon. On an average annual basis the amount transferred should be insignificant.

(b) Live Coal Stockpile and Handling Plant

The live coal stock is contained in a 160 m long concrete silo located below ground between the reserve coal pile and the power plant. The silo location can be seen in Figure 5-2 and a view of it under construction in 1978 is provided in Photo 5-6. The silo will hold 20,000 tonnes of coal, which will fuel 1 unit for 60 hours or 2 units for 30 hours. Coal handling is fully automated with washdown carried out on a periodic basis. Washdown water will be drained to transfer sumps which will also serve as collecting points for the weeping tile drains in the vicinity of the coal handling plant. The sumps will be designed in two sections such that any coal particles will be dropped out in the dirty section and the clean water will be pumped from the clean side of the sump. Coal from the dirty side will periodically be returned to the belt conveyors for discharge to the coal bunkers.

Water collected in the transfer sumps will be pumped into the ash lagoons at a maximum pumping rate of 190 l/s per unit. The average annual rate should not be more than 1 l/s.

(c) Coal Characteristics

Table 5-1 lists representative elemental analysis for the Coronach area lignite to be burned in the Poplar River station. These representative values were compiled through examination and evaluation of a number of separate laboratory analyses. Some of the analyses were done prior to the Board study and some were initiated by and completed for the Plant, Mine and Reservoir Operations Committee during the study. The analyses considered in arriving at the representative values are as follows:

1. Saskatchewan Research Council (M. Petrow) - Analysis of 62 primary Rockglen samples that are within the 35 year mining plan area.
2. Saskatchewan Research Council - Analysis of 8 coal samples - 1973.
3. Saskatchewan Research Council - Analysis of 8 coal samples - 1975. (These split from SRC 1973 samples.)

PHOTO 5-5 EVAPORATION POND DURING POWER PLANT CONSTRUCTION IN MAY, 1978



PHOTO 5-6 COAL HANDLING SILO DURING CONSTRUCTION IN MAY, 1978



TABLE 5-1 ELEMENTAL ANALYSIS - CORONACH AREA LIGNITE

<u>CONSTITUENT</u>	<u>NO. OF SAMPLES</u>	<u>REPRESENTATIVE CONCENTRATION</u>
Aluminum	10	1.7
Calcium	10	1.8
Iron	10	0.62
Magnesium	10	0.45
Potassium	9	0.39
Silicon	9	4.0
Sodium	9	0.23
Sulphur		0.80
Titanium	9	0.051
<u>ppm</u>		
Antimony	3	1.1
Arsenic	11	6.8
Barium	3	465
Beryllium	2	<0.16
Bismuth	1	9.0
Boron	11	37*
Bromine	3	2.7
Cadmium	10	<0.6
Chromium	12	25
Cobalt	11	9.3
Copper	12	46
Fluorine	11	86
Lead	11	<29
Lithium	8	9.0
Manganese	12	130
Mercury	10	0.32
Molybdenum	1	9.0
Nickel	11	37
Phosphorus	8	0.152
Selenium	11	0.8
Strontium	12	197
Thallium	2	<0.5
Uranium	11	7.9
Vanadium	11	23
Zinc	11	40
Zirconium	2	80

\*If the eight very low boron values obtained by SRC (Annex B) are not used in the arithmetic mean computation, the mean for boron is 139 ppm.

4. A-A-Detectametal - Analysis of 8 coal samples - 1975. (These split from SRC 1973 samples.)
5. State of Montana - Analysis of 8 coal samples - 1975. (These split from SRC 1973 samples.)
6. Silverspoon Research and Consulting - Analytical Results on 45 coal samples.
7. Western Research and Development - Analysis of coal samples from the 1975 Boundary Dam Test Burn - 1975.
8. Chemical and Geological Lab - Analysis of coal samples from the 1975 Boundary Dam Test Burn - 1975.
9. CDM/ACCU Labs - Analysis of coal sample from the 1975 Boundary Dam Test Burn - 1978.
10. EPA Corvallis Laboratory - Analysis of coal sample from the 1975 Boundary Dam Test Burn - 1978.
11. EPA Athens Laboratory - Analysis of coal sample from the 1975 Boundary Dam Test Burn - 1978.
12. U.S. Department of Energy Coal Laboratory - Analysis of lignite used in the 1975 Boundary Dam Test Burn.

The data in Table 5-1 represents the Plant, Mine and Reservoir Operations Committee's best estimate of average coal characteristics based on the information available. Details of the test data used are contained in Annex B.

#### 5.1.13 Ash Disposal System

##### (a) Ash Quantities and Characteristics

The maximum coal consumption rate is estimated to be 249,500 kg/hr (550,060 lb/hr). This is based on a boiler heat input of  $3000 \times 10^6$  BTU/hr and an "as received" coal calorific value of 5,454 BTU/lb.

To determine monthly coal consumption and ash production rates, the power plant is assumed to operate as follows:

With 1 unit

- 1 month per year maintenance shutdown (June)
- 87 percent capacity operation for 11 months

With 2 units

- 1 month per year maintenance shutdown for each unit (1 down in June, 1 down in August)
- 87 percent capacity operation of 2 units for 10 months and 1 unit for 2 months each year

Each of these operational methods yields an annual station capacity factor of approximately 80 percent. Table 5-2 gives monthly coal consumption figures based on the above assumed plant operation and monthly ash production based on 13.6 percent ash content for "as received" coal. In computing the figures, all months were assumed to be 30.44 days (365.25/12) for convenience of computation.

Table 5-3 lists representative constituent concentrations for the Poplar River ash. These were determined by interpretation and evaluation of a number of laboratory analysis of ash samples, details of which are provided in Annex B. The various sources of information considered in arriving at the representative values are as follows:

1. Saskatchewan Research Council (M. Petrow) - Analysis of 62 primary Rockglen samples that are within the 35 year mining plan area.
2. Saskatchewan Research Council - Analysis of 8 ash samples - 1973.
3. Saskatchewan Research Council - Analysis of 8 ash samples - 1975.
4. A-A-Detectametal - Analysis of 8 ash samples - 1975. (These split from SRC 1973 samples.)
5. Western Research and Development - Analysis of ash samples from the 1975 Boundary Dam Test Burn - 1975.
6. Chemical and Geological Lab - Analysis of ash samples from the 1975 Boundary Dam Test Burn - 1975.
7. CDM/ACCU Labs - Analysis of bottom, dust collector and upper ash from the 1975 Boundary Dam Test Burn - 1978.
8. EPA Corvallis Laboratory - Analysis of bottom, dust collector and upper ash from the 1975 Boundary Dam Test Burn - 1978.
9. EPA Athens Laboratory - Analysis of bottom, dust collector and upper ash from the 1975 Boundary Dan Test Burn - 1978.

TABLE 5-2

SUMMARY - COAL CONSUMPTION AND ASH PRODUCTION

MONTH	1 UNIT		2 UNITS	
	COAL	ASH	COAL	ASH
	Consumption (kgx10 <sup>6</sup> )	Production (kgx10 <sup>6</sup> )	Consumption (kgx10 <sup>6</sup> )	Production (kgx10 <sup>6</sup> )
Jan	158.57	21.56	317.14	43.12
Feb	158.57	21.56	317.14	43.12
Mar	158.57	21.56	317.14	43.12
Apr	158.57	21.56	317.14	43.12
May	158.57	21.56	317.14	43.12
June	0.00	0.00	158.57	21.56
July	158.57	21.56	317.14	43.12
Aug	158.57	21.56	158.57	21.56
Sept	158.57	21.56	317.14	43.12
Oct	158.57	21.56	317.14	43.12
Nov	158.57	21.56	317.14	43.12
Dec	158.57	21.56	317.14	43.12
TOTAL	1744.27	237.16	3488.54	474.32

( 1 lb = 0.45359 kg, 1 kg = 2.2045 lb)

TABLE 5-3

CONSTITUENT CONCENTRATIONS - POPLAR RIVER ASH

<u>CONSTITUENT</u>	<u>CONCENTRATION</u>
<u>Major</u>	
$\text{SiO}_2$ (%)	50.0
$\text{Al}_2\text{O}_3$ (%)	17.4
$\text{TiO}_2$ (%)	1.3
$\text{Fe}_2\text{O}_3$ (%)	7.5
$\text{Na}_2\text{O}$ (%)	1.7
$\text{K}_2\text{O}$ (%)	1.6
$\text{CaO}$ (%)	13.9
$\text{MgO}$ (%)	5.1
$\text{P}_2\text{O}_5$ (%)	0.35
$\text{SO}_3$ (%)	1.20
<u>Minor and Trace Element</u>	
Antimony (ppm)	2.1
Arsenic (ppm)	18.5
Barium (%)	0.53
Beryllium (ppm)	2.8
Boron (ppm)	425
Bromine (ppm)	1.9
Cadmium (ppm) <sup>1</sup>	1.8
Cerium (ppm)	114
Cesium (ppm)	3.9
Chlorine (ppm)	463
Chromium (ppm)	60
Cobalt (ppm)	9.8
Copper (ppm)	43
Fluorine (ppm)	94
Gallium (ppm)	1,089
Gold (ppb)	4.6
Hafnium (ppm)	12
Holmium (ppm)	6.2
Lead (ppm)	30
Manganese (ppm)	408
Molybdenum (ppm)	17
Nickel (ppm)	39
Rubidium (ppm)	41
Selenium (ppm)	<2.6
Silver (ppm)	0.38
Strontium (ppm)	2,053
Tantalum (ppm)	1.7
Tellurium (ppm)	94
Thorium (ppm)	14
Tungsten (ppm)	2.5
Uranium (ppm)	11
Vanadium (ppm)	74
Zinc (ppm)	21
Zirconium (ppm)	238

The data in Table 5-3 represents the Plant, Mine and Reservoir Operations Committee's best estimate of major and minor constituents and trace elements in the Poplar River coal ash (composite of bottom ash and flyash) based on the information available.

(b) Ash Collection and Transport

Bottom ash, which averages about 23.5 percent of the total ash, is collected wet in the bottom ash hopper located beneath the boiler. Flyash, representing the other 76.5 percent, is dry collected by a 99.5 percent efficient electrostatic precipitator. Both the bottom and flyash are transported to the ash slurry tank from where they are pumped to the ash lagoon in a combined bottom ash/flyash slurry.

The ash transport is accomplished with a high pressure and a low pressure pump for flyash and bottom ash respectively, each having a capacity of 170 l/s (2700 U.S. gpm). These are backed up by two similar standby pumps. Operating the bottom ash and flyash pumps alternately, it is expected that ash will be slurried at the 170 l/s rate for a total of about 6 hours out of each 8-hour shift giving an average slurry water flow rate of approximately 130 l/s. Using the ash production figures of  $21.56 \times 10^6$  kg per month given in Table 5-2, the ash to water ratio is 63,060 mg/l, which is a 6.3% slurry. Taking into account all other wastewater discharges into the ash lagoon from the plant, an additional 42.3 l/s for one unit operation, the total water input prior to loss from evaporation and seepage becomes 172.3 l/s. The ash to water ratio in the lagoon then becomes 47,580 mg/l.

(c) Ash Lagoons

Location and Design

Seven ash lagoon cells are proposed in the area southeast of the plant. Photo 5-7 shows the rolling topography of the area. The approximate layout of the lagoons is shown in Figure 5-1. The exact location of the lagoon boundaries has been finalized for lagoons 1, 2 and 3 only. The location and sizes indicated for the other lagoons are assumed based on the land already purchased and will be used as representative until final information is received. If constructed as shown, however, the southern boundary of lagoon cells 6 and 7 would be approximately 200 m south of the right wing dike of Morrison Dam. The lagoon cells are arranged in two parallel rows with a 75 m wide clear water return ditch between the rows.

The lagoons will be constructed by leveling a portion of the interior land surface and diking around the perimeter. The perimeter dikes for the ash lagoon area will be a zoned section with an impervious core and upstream blanket, and a common fill shell. Interior dikes between ash lagoon cells will be a homogeneous common fill section.

PHOTO 5-7 VIEW LOOKING SOUTHWARD FROM THE STATION OF THE  
AREA TO BE OCCUPIED BY THE ASH LAGOONS - MAY, 1978



Table 5-4 lists pertinent physical dimensions of each lagoon. The data for lagoon cells 1, 2 and 3 are based on the preliminary designs by SPC whereas the other figures are assumed dimensions. The full level volume figures given are for an ash surface level 2 m below the dike crest.

TABLE 5-4 ASH LAGOON STATISTICS

LAGOON NUMBER	BOUNDARIES ON SITE		BASE ELEV. (m)	DIKE ELEV. (m)	FULL ELEV. (m)	MAXIMUM	
	GRID (FIGURE 5-1) N-S (m)	E-W (m)				SURFACE AREA (m <sup>2</sup> )	USEABLE VOLUME (m <sup>3</sup> )
1	2200-2800	4075-4822	757.0	764.0	762.0	391,000	2,162,000
2	1662-2200	4075-4822	759.0	766.2	764.2	402,000	2,456,000
3	1662-2570	3075-4000	758.0	763.3	761.3	586,000	2,429,000
4	860-1662	3250-4000	756.0	763.3	761.3	602,000	3,781,000
5	860-1662	4075-4822	762.0	766.2	764.2	599,000	1,973,000
6	50-860	4075-4822	768.0	772.0	770.0	605,000	2,442,500
7	50-860	3250-4000	762.0	766.2	764.2	608,000	2,170,000

NOTE: Firm design estimates available for lagoon numbers 1-3 only.

Elevation-storage tables for the lagoons are given in Annex H. The base elevations shown on these curves are assumed elevations to which the lagoon bottoms will be levelled. The profiles in Figure 5-10 show the natural ground surface elevation along each side of the two rows of lagoons. Figure 5-11 shows the position and elevation of lagoon 3 relative to the cooling water discharge channel. The location of this profile is along Section F-F shown in Figure 5-1.

#### Subsurface Conditions

The subsurface conditions have been determined by a 14 hole drilling program in the ash lagoon area during the summer of 1978 (Ground Engineering, 1978). The information obtained from the program is supplemented by previously complete test holes immediately north, southwest and southeast of the ash disposal area. The location of all test holes is shown in Figure 5-1.

A subsurface profile, estimated from the test hole information, is shown in Figure 5-12. The profile is located along an assumed seepage path for water seeping from lagoons 1,2 or 3. As shown in Figure 5-1, the profile runs from the northwest corner of lagoon 1 to the East Poplar River below Morrison Dam.

FIGURE 5-10 ASH LAGOON PROFILES

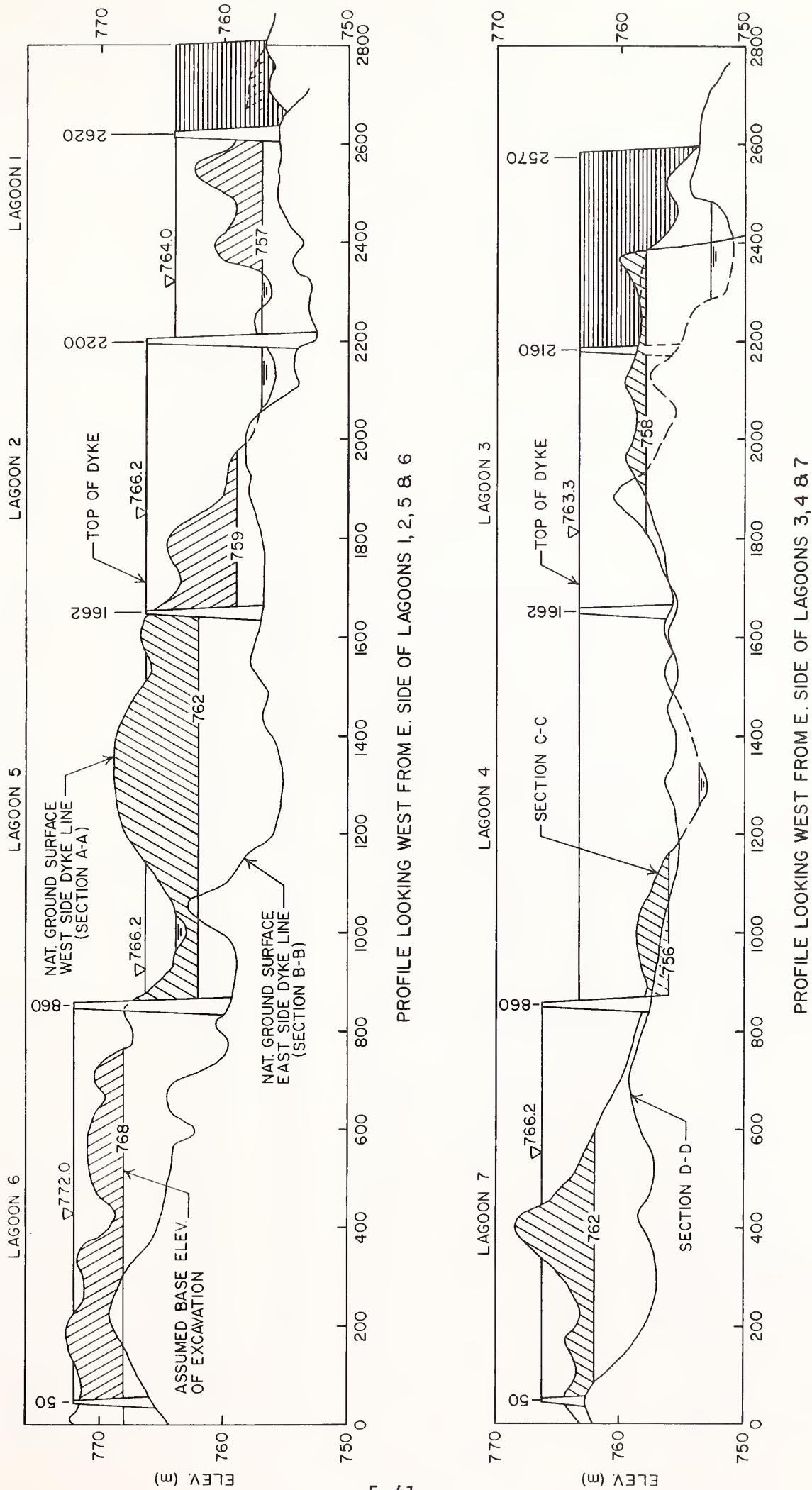


FIGURE 5-11 SECTION F-F OF ASH LAGOON DYKE AND C.W. DISCHARGE CHANNEL

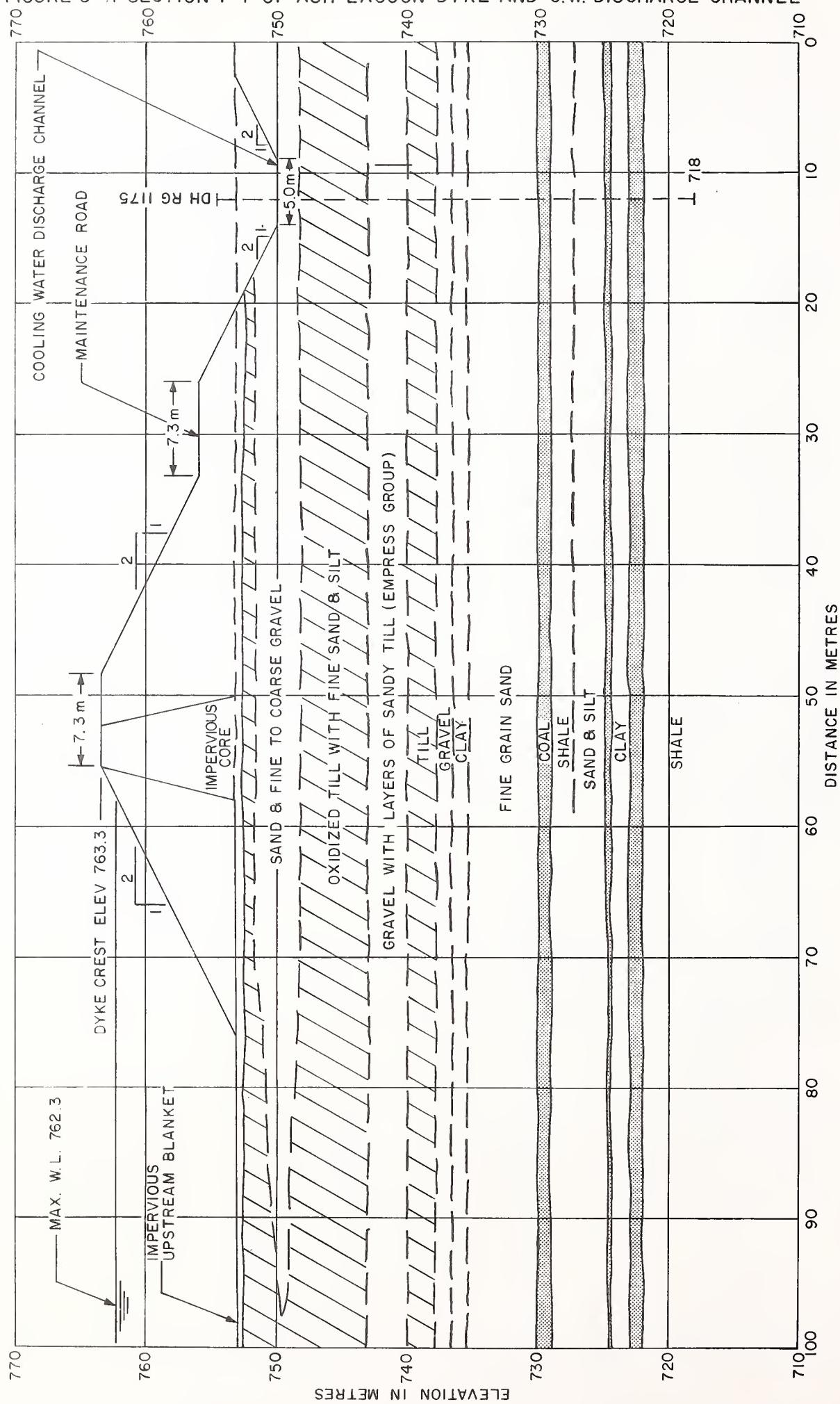
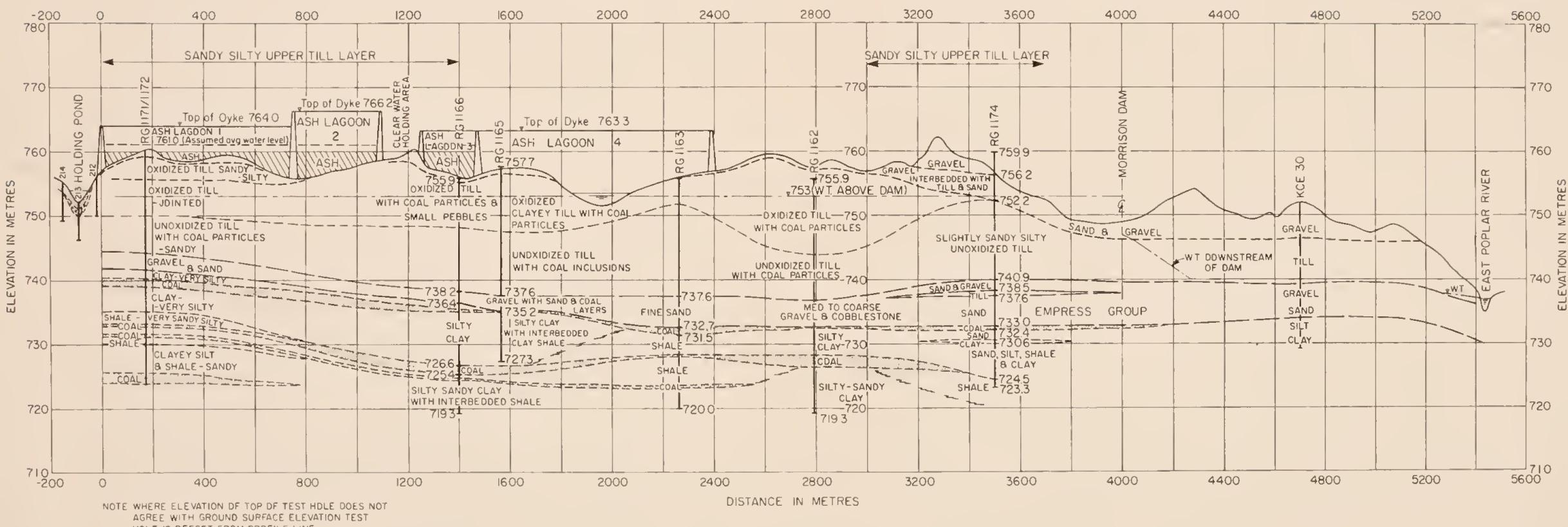


FIGURE 5-12: SUBSURFACE PROFILE ALONG PROBABLE SEEPAGE PATH FROM ASH LAGOONS (SECTION E-E, FIGURE 5-1)





The subsurface profile shows a thin (approximately 1/2 m average) layer of surficial sands and gravels over most of the ash lagoon area. These surficial granular deposits deepen in some of the higher hills and knolls, and reach depths varying from 4 to 8 m near the southeast corner of the lagoon area. Judging from test hole KCE30 near the river on the centreline of the right wing dike of Morrison Dam, the surface layer may persist at depths varying from 2 to 8 m from the southeast corner of the lagoons to the river below Morrison Dam although this has not been verified.

Below the surficial layer of sands and gravels is a layer of till 14 to 20 m deep in the lagoon area and decreasing in depth to 6 to 8 m downstream of the lagoons where the surficial deposits are deeper. For the most part, the upper half of the till layer is oxidized and the lower half unoxidized. At all drill hole locations, the till was found to be highly fractured and jointed. It is also interspersed with coal particles and small pebbles and in two general areas exhibits a sandy silty tendency. These areas appear to coincide with higher elevation surface formations. Their estimated limits are delineated in Figure 5-1. One area is an east-west oriented strip across the northern end of the lagoons encompassing lagoon 1 and the northern portions of lagoons 2 and 3. The other is across the southern end of the lagoons extending eastward to Morrison Dam and may be part of a larger general area to the southeast of the lagoons. For the most part, the till in these areas is only slightly sandy silty or the sandy silty character is confined to the upper 3 to 6 m. This is particularly true along the western edge of the lagoon area. However, drill hole RG 1164, located east of the lagoons between the lagoon area and the East Poplar River, shows the middle zone of the till being quite silty with some very fine sand. Drill hole RG 1175 at the northern edge of the lagoons shows a very inconsistent till layer with interbedded gravel and sand and generally very silty consistency throughout. This would suggest that the east - west strip of slightly sandy silty till across the northern end of the lagoons could contain localized areas of higher permeability leading down to the underlying sands and gravels.

The layer of sands and gravels (Empress Group) underlying the upper till layer varies in thickness from 2 to 8 m and remains uniformly horizontal throughout between elevation 732 m and 744 m, with a slight rising tendency to the northern end of the ash lagoon area. The granular material has a consistency ranging from fine sand to coarse gravel and cobblestones. It probably represents the main aquifer for groundwater movement around Morrison Dam from the reservoir to the river downstream.

Beneath the Empress Group sands and gravels are interbedded layers of coal, shale and silty clay of the Ravenscrag Formation to beyond the depth of the drill holes, most of which terminate approximately at elevation 720 m.

## Seepage Control

Falling head permeability tests (Ground Engineering, 1978) have indicated that the coefficient of permeability of the in-situ till is in the range of  $9 \times 10^{-5}$  to  $5.2 \times 10^{-5}$  cm/s. The SPC has proposed to reduce the permeability to a maximum of  $1.5 \times 10^{-5}$  cm/sec (or 4.7 m/year) by re-working and compacting the upper 0.6 m of the lagoon base and suggest that imperviousness will further improve with settling of ash. The use of PVC or rubber lining over the base of the lagoon is not now being considered because of the high cost.

## Operational Procedure

Current plans call for the operation of three lagoons in sequence. One will be filled until the water reaches a predetermined level. It is assumed that this level would be 1 m below the dike crest elevation. The lagoon will then be left to settle for 1 month followed by a 1 month decant period when the excess water above the settled ash will be released back into the reservoir via the clear water holding area and the cooling water discharge channel. The ash lagoon discharge structure will consist of a floating skimming boom to retain floating substances, a bottom weir or retaining wall that can be adjusted to the level of ash being retained and a water outlet and control system that can be adjusted to the required level of discharge.

During the settling and decanting period for the first lagoon, ash will be slurried sequentially into the second and third lagoons and continued until their predetermined water levels are reached. This procedure of operating three lagoons in series will be continued until all three are full of ash. The next set of three lagoons will then be brought into service. Except for lagoons 1, 2 and 3 for which preliminary designs are available, the order of filling implied by the lagoon numbers in Figure 5-1 is an assumed order only.

Approximate fill times for the ash lagoons have been computed for 1 and 2 unit operation assuming that the maximum ash surface level that could be attained in each lagoon is 2 m below the dike crest. The volumes corresponding to these levels are given in Table 5-4 for each lagoon. The ash volumes used in the computations are based on a wet ash density of 1,200 kg/m<sup>3</sup> (74.9 lb/ft<sup>3</sup>). This yields the following volumetric ash production figures.

$$1 \text{ unit at } 237.16 \times 10^6 \text{ kg/year} = 197,630 \text{ m}^3/\text{yr.}$$

$$2 \text{ units at } 474.32 \times 10^6 \text{ kg/year} = 395,270 \text{ m}^3/\text{yr.}$$

The 1 and 2 unit life spans for the ash lagoons computed using these volumetric production rates are as follows:

	Ash volume (m <sup>3</sup> )	Life Span (Years)	
		1 Unit	2 Units
Lagoons 1, 2 & 3	7,047,000	35.7	17.8
Lagoons 4, 5 & 6	8,196,500	41.5	20.7

## 5.2 Inputs and Losses to Surface Water Systems

### 5.2.1 Station Water Balance

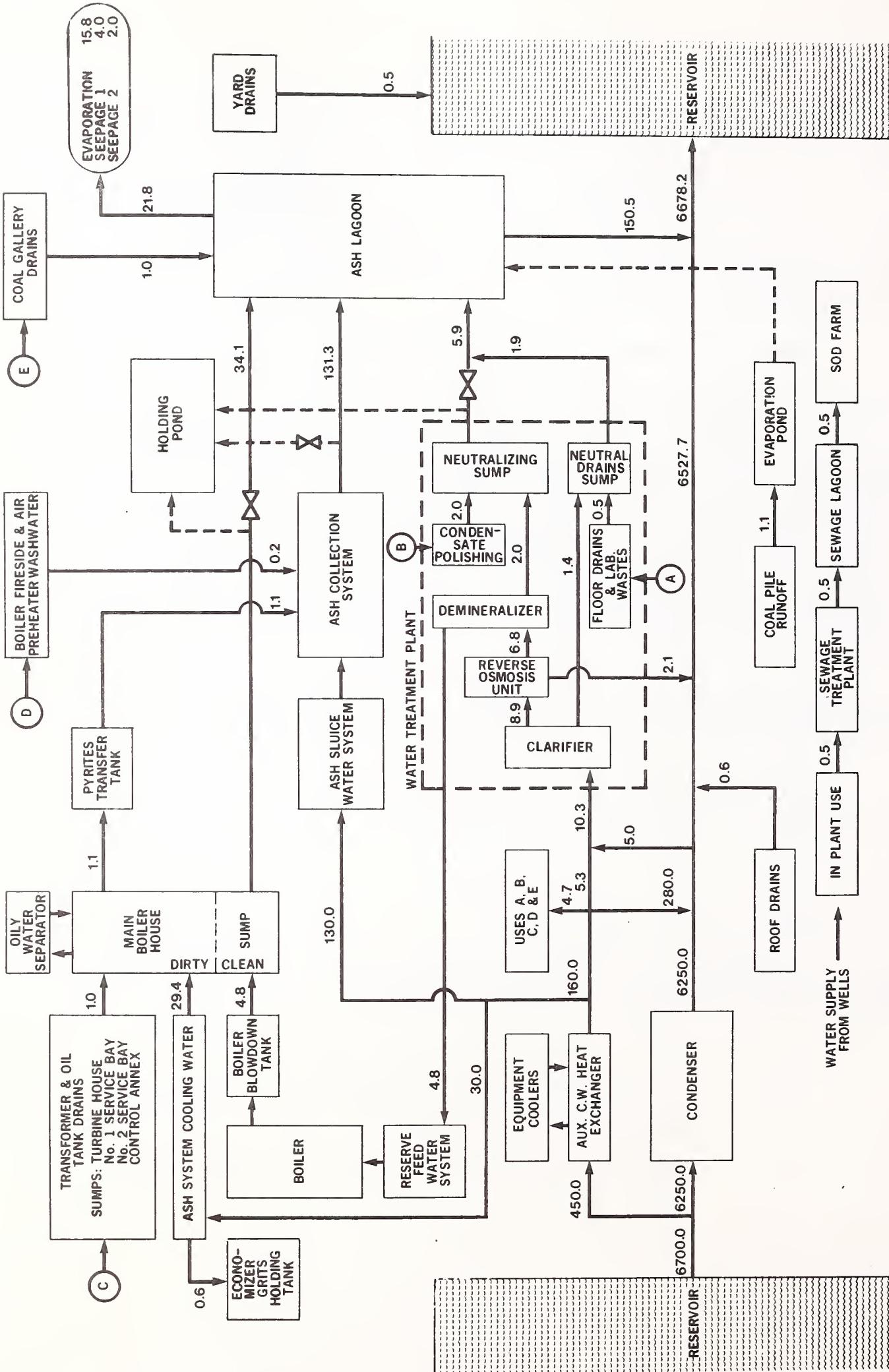
The estimated annual average water balance for the Poplar River generating station is summarized in Figures 5-13 and 5-14 for the 1 unit and 2 unit cases respectively. Corresponding figures plus additional information on maximum flow rates and annual volumes are given in Tables 5-5 and 5-6. There have been numerous wastewater management system outlines provided by the SPC during the course of this study, with Issue 6 (Saskmont, Oct., 1978) containing some of the most recent changes. Although the system outlined here should fairly well represent the actual wastewater flow rates and distribution, further modifications may have been made since obtaining the most recent information from the SPC.

The water balance diagrams show direct waste releases to the plant cooling water discharge, and hence to the reservoir, from 4 sources only:

- (i) auxiliary cooling water system
- (ii) plant roof and yard drains
- (iii) reverse osmosis portion of the water treatment plant
- (iv) ash lagoons.

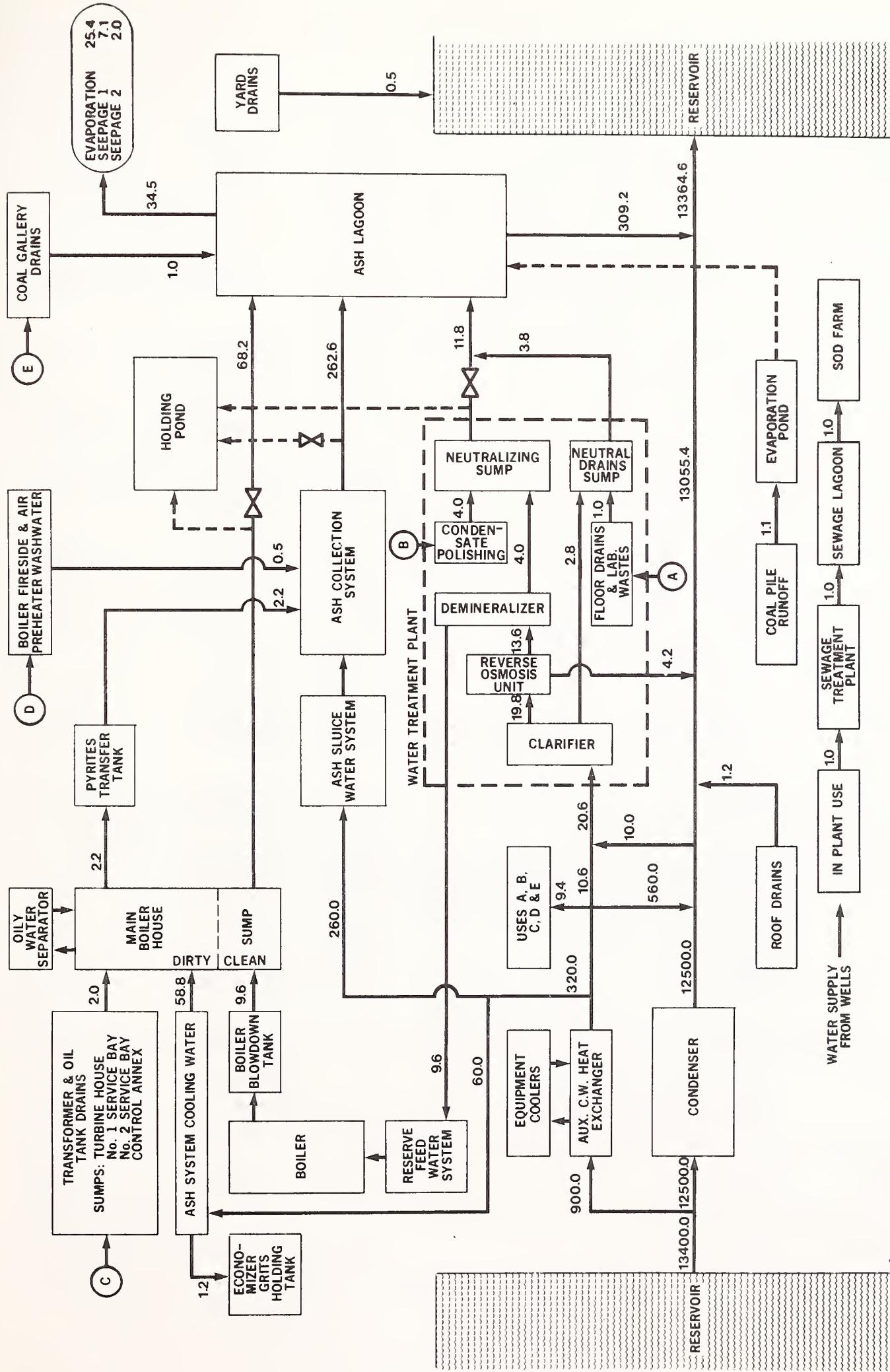
A number of wastewater sources are directed to the ash lagoon in addition to the ash slurry, and hence contribute to the ash lagoon water quality. These can be summarized as follows:

- (i) wastes from the main boiler house sump
- (ii) wastes from the water treatment plant neutralizing sump and neutral drains sump
- (iii) occasional wastewater from boiler fireside and air pre-heater washing
- (iv) coal gallery drains



Note: All flow rates are litres/sec.

FIGURE 5-13 POPLAR RIVER G. S. WATER BALANCE - 1 UNIT



Note: All flow rates are litres/sec.

FIGURE 5-14 POPIA RIVER G. S. WATER BALANCE : 2 UNITS

TABLE 5-5 POPLAR RIVER G.S. WATER BALANCE - 1 UNIT

EFFLUENT DESTINATION	SOURCE OF EFFLUENT OR LOSS	MAXIMUM FLOW (l/s)	ANNUAL VOLUME (M <sup>3</sup> )	AVERAGE FLOW (l/s)
MAIN BOILER HOUSE SUMP	Sumps: Turbine House No. 1 Service Bay No. 2 Service Bay Control Annex Ash System Cooling Water Boiler Blowdown	36.0 36.0 36.0 36.0 30.0 160.0	31,550 0 0 0 927,670 151,460	0 1.00 0 0 29.40 4.80
	<b>TOTAL</b>		<b>1,110,610</b>	<b>35.20</b>
ASH LAGOONS	Coal Gallery Drains Main Boiler House Sump Water Treatment Plant Neutralizing Sump Neutral Drains Sump Ash Slurry Pyrites Transfer Tank Boiler Fireside & Air Preheater Wash Losses From Ash Lagoons Evaporation Seepage to E. Poplar River Seepage to C.W. Channel	190.0 180.0 170.0 360.0 -32.0	31,550 1,075,970 4,101,930 34,710 7,850 -500,000 -126,210 - 63,110	1.00 34.10 130.00 1.10 0.25 -15.85 -4.00 -2.00
	<b>TOTAL</b>		<b>4,748,850</b>	<b>150.50</b>
RESERVOIR	Condenser Discharge Auxiliary Cooling Water Return Used in Water Treatment Plant Reverse Osmosis Rejects Ash Lagoon Decant Roof Drains Yard Drains	8,564.0 450.0 -157,770 66,260 4,748,850 47.3 15,780	197,208,000 8,834,920 -5.00 150.50 18,930 0.60 0.50	6,250.00 280.00 -5.00 2.10 150.50 6,678.70
HOLDING POND	Main Boiler House Sump Water Treatment Plant Ash Collection System		Occasional Releases	
SEWAGE LAGOON	Showers, Washrooms, Etc.	1.8	16,440	0.52
EVAPORATION POND	Coal Pile Runoff		35,000	1.10

sec/yr 31,553,280

TABLE 5-6 POPLAR RIVER G.S. WATER BALANCE - 2 UNITS

EFFLUENT DESTINATION	SOURCE OF EFFLUENT OR LOSS	MAXIMUM FLOW (l/s)	ANNUAL VOLUME (M <sup>3</sup> )	AVERAGE FLOW (l/s)
MAIN BOILER HOUSE SUMP	Sumps:   Turbine House No. 1 Service Bay No. 2 Service Bay Control Annex Ash System Cooling Water Boiler Blowdown	72.00 72.00 72.00 72.00 60.00 320.00	63,110 1,855,330 302,910	2.00 58.80 9.60
	<b>TOTAL</b>		<b>2,221,350</b>	<b>70.40</b>
ASH LAGOONS	Coal Gallery Drains Main Boiler House Sump Water Treatment Plant Neutralizing Sump Neutral Drains Sump Ash Slurry Pyrites Transfer Tank Boiler Fireside & Air Preheater Wash Losses From Ash Lagoons Evaporation Seepage to E. Poplar River Seepage to C.W. Channel	190.00 360.00  320.00 720.00  -57.00	31,550 2,151,930 252,430 119,900 8,203,850 69,420 15,720  -800,000 -224,030 - 63,110	1.00 68.20 8.00 3.80 260.00 2.20 0.50  -25.36 - 7.10 - 2.00
	<b>TOTAL</b>		<b>9,757,660</b>	<b>309.24</b>
RESERVOIR	Condenser Discharge Auxiliary Cooling Water Return Used in Water Treatment Plant Reverse Osmosis Rejects Ash Lagoon Decant Roof Drains Yard Drains	17,128.00 900.00  132,520 9,757,660 37,860 15,780	394,416,000 17,669,840 -315,530 4.20 309.24 1.20 0.50	12,500.00 560.00 -10.00  421,714,130 13,365.14
HOLDING POND	Main Boiler House Sump Water Treatment Plant Ash Collection System		Occasional Releases	
SEWAGE LAGOON	Showers, Washrooms, Etc.		32,800	1.04
EVAPORATION POND	Coal Pile Runoff		35,000	1.10

sec/yr 31,553,280

- (v) occasional releases of excess water from the evaporation pond
- (vi) occasional releases of treated water from the holding pond.

The ash slurry itself periodically contains the wastewater from the pyrites transfer tank discharge.

The remainder of this section of the report deals first with the cooling water system and then sequentially with the four effluent sources that release directly to the cooling water.

#### 5.2.2 Condenser Cooling Water Discharge

##### (a) Flow Rate and Temperature

It is estimated that during 9 to 10 months of the year reservoir water temperatures will be sufficiently low to permit the operation of only one circulating water pump per unit. The standby pump will be used during the 2 or 3 months of warmer summer temperatures. The cooling water intake flows in Figures 5-13 and 5-14 are based on 1 pump per unit operating at the reservoir full supply level of 753 m. This is somewhat higher than what the average operating level is likely to be.

Based on these assumptions, the cooling water intake rates are:

for 1 unit        6,700 l/s

for 2 units       13,400 l/s

An average flow of 450 l/s per unit is drawn from the cooling water system prior to the condenser leaving a condenser flow of 6,250 l/s per unit. The only direct additions to this water in the condenser are temperature and possibly chlorine.

With one pump operating per unit, condenser flows and temperature rise were computed for different reservoir levels. These are based on a 12.39°C rise with the design flow of 7,635 l/s. All flow rates have been reduced by 450 l/s to account for the auxiliary cooling water flow extracted prior to the condenser. The results are as follows:

Reservoir Level (m)	Flow Per Unit (l/s)	Condenser Temp. Rise (°C)
753	6,250	15.1
752	6,050	15.6
751	5,800	16.3
750	5,550	17.0
749	5,250	18.0
748	4,950	19.1
747	4,600	20.6
746	4,150	22.8
745	3,550	26.6

With higher reservoir temperatures of 20°C and 25°C, the standby pump in operation and flows throttled (T) when necessary to the condenser capacity limit (17,128 l/s for two units), the flow rates, temperature rise and discharge temperature with two units operating would be approximately as listed below:

Reservoir Level (m)	No. of Pumps	Condenser Flow (l/s)	Condenser Temp. Rise (C°)	Condenser Discharge Temp. (°C)	
				20°C Inlet	25°C Inlet
753	3(T)	17,128	11.0	31.0	36.0
752	3(T)	17,128	11.0	31.0	36.0
751	3(T)	17,128	11.0	31.0	36.0
750	3	17,100	11.1	31.1	36.1
749	3	16,200	11.7	31.7	36.7
748	3	15,300	12.4	31.4	37.4
747	3	14,250	13.3	33.3	38.3
746	3	12,900	14.7	34.7	39.7
745	3	11,100	17.0	37.0	42.0

These calculations are approximate in that they assume a constant heat rejection rate. They are, however, representative of the conditions that could occur with different reservoir water levels. Maximum inlet water temperatures in excess of 25°C have been observed at the Boundary Dam generating station and can be expected at Poplar River with the smaller, shallower reservoir.

1

As the table shows, water being discharged from the condenser will have a temperature exceeding 36°C for all reservoir levels if the inlet temperature is greater than 25°C. The discharge temperature gradually increases as the reservoir level drops below 750 m. A drop in temperature of 2 or 3°C can be expected along the 3,800 m discharge channel resulting in water at 33°C or higher being discharged into the reservoir.

(b) Condenser Anti-fouling

The proposed chlorine application rate of 5 mg/l of Cl<sub>2</sub> for three 10 minute periods per day constitutes usage rates of:

$$\begin{aligned}1 \text{ unit} & \quad 6,050 \times 30 \times 60 \times 5 \text{mg/l} = 54.45 \text{ kg/day} \\& \quad 11/12 \times 365 \times 54.54 = 18,200 \text{ kg/year} \\& \quad 18,200 /11 = 1,650 \text{ kg/month} \\2 \text{ units} & \quad 108.90 \text{ kg/day} \\& \quad 36,400 \text{ kg/year} \\& \quad 3,300 \text{ kg/month}\end{aligned}$$

This is based on a constant application and flow rate for 11 months of the year and a 1 month maintenance outage.

Attempts to keep the chlorine residual down to only those levels (0.5 mg/l or less) needed to remove biological growth in the condenser would minimize chloride addition to the reservoir from this source. It may be that chlorine will not be required during all months of the year and the annual usage figures could be less. On the other hand, the need may arise for a higher dose rate or longer duration of application. Also, with a given concentration, the variable cooling water flow rate would cause the chlorine usage to vary. The values listed should, however, be good approximations of the average use.

Through reaction with the hydrochloric acid formed when chlorine is added to water, a certain amount of alkalinity is consumed(White, 1972). The alkalinity removed per kilogram of Cl<sub>2</sub> added is computed to be 0.643 kg (as CaCO<sub>3</sub>) and remains fairly constant due to the high dissociation rate of hydrochloric acid. Alkalinity reduction in Cookson Reservoir based on the chlorine additions given above would be:

$$\begin{aligned}1 \text{ unit} & \quad 1,060 \text{ kg/month} \\2 \text{ units} & \quad 2,120 \text{ kg/month}\end{aligned}$$

5.2.3 Auxiliary Cooling Water Discharge

The portion of the 450 l/s per unit total auxiliary cooling water flow not reused for ash slurry in the plant is released directly to the main cooling water discharge. The release rates are expected to be:

$$\begin{aligned}1 \text{ unit} & \quad 280 \text{ l/s} \\2 \text{ units} & \quad 560 \text{ l/s}\end{aligned}$$

The quality of the released water is expected, on average, to be essentially the same as that of the reservoir with the exception of a temperature elevated about 5°C. This is lower than the temperature rise for the main cooling water stream and will contribute in a minor way to higher temperatures in the reservoir. The addition of oils or grease to the auxiliary cooling water from leaks in equipment coolers is unlikely since the secondary closed loop system prevents direct contact of the once-through auxiliary cooling water with the equipment being cooled.

#### 5.2.4 Plant Roof and Yard Drains

The roof and yard drains could constitute a loss to the surface water system in that they replace flow which would have run off from the site area prior to construction. However, the pre-construction runoff would probably be less than 10 percent of runoff after construction from the roof areas and the graded site area. Hence, the loss to surface water should be assumed to be zero.

The average annual discharge to the reservoir from roof and yard drains, as listed on the water balance diagrams in Figures 5-13 and 5-14, is estimated to be:

1 unit                    1.07 l/s

2 units                    1.13 l/s

These quantities are based on an average annual rainfall of 0.355 m. The drains are designed to accommodate high concentration, short duration rainfalls. The quality of the drain water is expected generally to be as good as that of the reservoir water. However, unintentional releases of oil or other chemicals used on site could end up in the drains. Any such releases would then end up in the reservoir.

For the purpose of determining reservoir water quality on a monthly bases, plant roof and yard drains need not be considered as an input to the system.

#### 5.2.5 Reverse Osmosis Rejects

Rejects from the reverse osmosis (RO) plant will be sent directly to the cooling water discharge at the following rates:

1 unit                    2.1 l/s

2 units                    4.2 l/s

The RO unit is expected to reduce dissolved solids in the incoming reservoir water by more than 90 percent. This means that the 90 percent plus solids removed will be concentrated in the reject flow. RO membranes tend to reject calcium and magnesium among cations to a greater extent than sodium, and among anions, sulphate to a greater extent than chloride.

Since the reject flows are approximately 40 percent of the treated water used for boiler make-up, the concentration of salts in the reject water will be approximately 2.5 times the reservoir water concentration. With a considerable variation in the reservoir water concentrations, the only realistic way to determine the quality of the RO rejects is to compute the quality in the reservoir operation model on a monthly bases. Such a computation has been incorporated into the model (Section 8). An average quality of the reject water with an intake water TDS of 1280 mg/l is estimated as follows:

Element	Assumed		Quantity Added mg/l Kg/day(2 units)	Effluent Conc. mg/l
	Intake Conc.	mg/l		
Na	200		590	214.1
K	10		21	7.6
Ca	50		120	43.5
Mg	50		120	43.5
HCO <sub>3</sub>	450		900	326.6
CO <sub>3</sub>	0		0	0
SO <sub>4</sub>	500		1230	446.3
NO <sub>3</sub>	10		21	7.6
C1	20		42	15.2
SiO <sub>2</sub>	20		11	4.0
B	5		11	4.0
TDS	1280		3023	1097

These figures are based on rejection rates ranging from 84 to 98 percent, depending on the ion.

#### 5.2.6 Ash Lagoon Discharge

##### (a) Waste Inputs to Ash Lagoons

In order to characterize the water quality in the ash lagoons and the quality of seepage or decant, the quantity and quality of each of the separate waste inputs to the lagoons must be determined. Each of those inputs is individually dealt with below.

##### Ash Slurry

As indicated in Section 5.1.13, the ash slurry flow to the lagoons will average 130 l/s for 1 unit and 260 l/s for 2 units. This makes up about 75 percent of the flow to the lagoon.

The key element in determining the contribution that the ash slurry will make to the lagoon water quality is the amounts of various constituents that will be dissolved from the ash into the slurry water. In

order to estimate quantities of readily leachable material in the ash from the Poplar River station the following leachate tests were conducted:

- (i) static leaching with distilled water and a 1,000 mg/l ash to water ratio (Analysis by U. of Montana and Corvallis)
- (ii) static leaching with distilled water and a 100,000 mg/l ash to water ratio (Analysis by U. of Wisconsin)
- (iii) static leaching with water from Cookson Reservoir (June, 1978) and a 60,000 mg/l ash to water ratio (Analysis by U. of Montana)
- (iv) static leaching with water from Cookson Reservoir (June, 1978) doubled in concentration to better reflect reservoir water conditions that could occur and a 60,000 mg/l ash to water ratio (Analysis by U. of Montana).
- (v) static leaching with water from Cookson Reservoir (June, 1978), a 60,000 mg/l ash to water ratio and the ash heated (bottom ash to 2000°F, flyash to 325°F) before adding it to the test water (Analysis by U. of Montana).

The procedure used in conducting the leachate tests and the detailed results of the tests are contained in Annex E. The results have been summarized in Table 5-7. The figures listed are the quantities shown by the tests to be readily leachable. The results from the leachate tests with the 1,000 and 100,000 mg/l leaching solutions have been converted to reflect a 60,000 mg/l ash to water ratio, which approximates the slurry concentration expected in actual operation. The results from the remaining three tests were adjusted only by deducting the concentration of the various elements in the leaching water used for the tests. Thus all results reflect the concentrations that, according to the leachate tests, will be added to the slurry water from the ash.

It can be seen from Table 5-7 that the converted data from the 1000 mg/l tests shows very high values for Ca, Mg, HCO<sub>3</sub> and CO<sub>3</sub>. This is likely the result of a combination of using distilled water, a low concentration leachate solution and the conversion itself. The major portions of the added Ca and Mg initially in solution are expected to precipitate out as CaCO<sub>3</sub> and MgOH. Precipitation of Ca will happen more readily in the absence of an ice cover, allowing greater CO<sub>2</sub> uptake for the precipitation process. The leachate tests using reservoir water show an actual reduction in HCO<sub>3</sub> and, in some instances, Ca and Mg. It is realistic to assume that some reduction in HCO<sub>3</sub> will occur in the Poplar River ash lagoons under most conditions of operation, although with a lower alkalinity slurry water the reduction would likely be less than the 200 mg/l indicated. A HCO<sub>3</sub> reduction has been observed in the ash lagoons at Boundary Dam.

TABLE 5-7 RESULTS OF STATIC LEACHING TESTS ON POPLAR RIVER ASH

ELEMENT	CONCENTRATION (mg/l) ADDED TO ASH SLURRY WATER				REPRESENTATIVE VALUE BASED ON TESTS	
	LEACHATE TESTS		CONC. RES. WATER (60,000 mg/l)	RES. WATER & HEATED ASH (60,000 mg/l)		
	DIST. WATER (1000 mg/l) U. OF MONTANA	U. OF CORVALLIS (100,000 mg/l)				
Na	30	26.2	4.3	65	20	
K	9	1.6	1.1	1.2	0.9	
Ca	1005	123	0.4	3.2	-9.9	
Mg	90	<1.86	7.4	-2.9	-1.0	
B	5.8	5.1	15.5	15.7	9.3	
HCO <sub>3</sub> <sup>-</sup>	2099	-234	-262	-203	-200	
CO <sub>3</sub> <sup>=</sup>			56	30	40	
SO <sub>4</sub> <sup>=</sup>			236	148	180	
NO <sub>3</sub> <sup>-</sup>	96	236	2.2	-0.5	0	
Cl <sup>-</sup>		1.3	2.6	6.0	5	
SiO <sub>2</sub>	<5.76	3.2	4.1	3.9	4	
Cr	0.18	0.066	<0.10	<0.10	0.10	
Cd	0.007	<0.009	<0.02	<0.02	0.02	
Cu	0.035	0.12	<0.009	<0.40	0.10	
Fe	0.47	1.42	0.43	<0.35	0.35	
Mn	0.19	0.12	<0.016	<0.025	0.02	
Ni	0.058	<0.072	<0.005	<0.005	0.06	
Pb	0.089	<0.072	<0.020	<0.020	0.01	
Zn	0.035	0.30	0.013	<2.0	0.01	
Al		128	13.90	<2.0	15	
Sr		17.53	3.60		4	
V		0.53	0.047		0.05	
Mo		1.18	0.408		0.5	

The other element of interest in Table 5-7 is boron (B) which shows substantial pick up in all leachate tests. The values obtained when reservoir water was used show in excess of 10 mg/l of boron dissolving from the ash, 10 mg/l in the 60,000 mg/l ash leachate solution means that 166 mg of boron is dissolved from each gram of ash. This is 39 percent of the boron contained in the combined ash, which averages 425 mg/g. This can be compared to 34 percent for fly ash from Western low sulphur coal (Cox et al, 1978), 15.9 percent for a laboratory prepared ash from a Dunn County, North Dakota lignite (Somerville et al, 1978) and a range of 29 to 55 percent in a study of the Chalk Point and Four Corners power plants (Gladney et al, 1978). The 10 mg/l concentration added in the ash lagoon is also comparable to a 12 mg/l concentration in the ash pond effluent of a large coal-fired power plant at Fruitland, New Mexico (Dreesen et al, 1977). On the other hand, B concentrations in the ash lagoons for Units 3, 4 and 5 at the Boundary Dam station measured by the SPC and Environment Saskatchewan average only 1.5 mg/l. The lagoon for Unit 6 at Boundary Dam shows somewhat higher boron concentrations (values of 2.9, 3.2, 2.0, 2.3 and 5.1 recorded during the period March to November, 1978), which may be due to the increased efficiency in collection of the finer flyash particles.

#### Main Boiler House Sump Releases

The releases from the main boiler house sump are the next largest quantity input to the ash lagoons.

A small portion of this effluent comes from the dirty side of the sump via the pyrites transfer tank. The remainder is released from the clean side and goes direct to the ash lagoon. The average clean side, dirty side and total flow (l/s) from the main boiler house sump are as follows:

	<u>Clean side</u>	<u>Dirty Side</u>	<u>Total</u>
1 unit	34.1	1.1	35.2
2 units	68.2	2.2	70.4

The oily water separator will have removed oil contamination from the sump water before it is released. The clean side discharge on average should not be significantly lower in quality than the reservoir water. Approximately 83 percent of the water in the sump is ash system cooling water, which may pick up some salts and other elements from the ash, and 14 percent is boiler blowdown which, except for its ammonia content, is of much better quality than reservoir water. With the provision to release abnormally contaminated water to the holding pond, it is reasonable to assume that the TDS of releases to the ash lagoon from the clean side of the sump will be approximately the same as reservoir TDS levels. The estimated ammonia level of 2.6 mg/l in the boiler blowdown would add about 0.4 mg/l ammonia to the total sump release.

The dirty side discharge, transported to the ash lagoons via the pyrites tank, is of sufficiently low volume that it will contribute negligibly to the ash lagoon water quality.

## Water Treatment Plant

The input to the ash lagoons from the water treatment plant will originate from both the neutral drains sump and the neutralizing sump. These wastes will total 5.9 l/s for 1 unit and 11.8 l/s for 2 units. They will contain chemicals added to the clarifier and used to regenerate the ion exchange resins in the demineralizer and condensate polishing units. Based on the estimated annual chemical usage in Section 5.1.4, an assumed average intake water quality and the amount of material removed by the R.O. plant prior to the demineralizer, the discharge to the ash lagoons will contain these average concentrations:

Element	Concentrations (mg/l)		
	Intake Water	Added	Effluent
Na	200	52	252
K	10	1	11
Ca	50	1	51
Mg	50	1	51
HCO <sub>3</sub>	450	72	522
CO <sub>3</sub>	0	0	0
SO <sub>4</sub>	500	153	653
NO <sub>3</sub>	10	1	11
C1	20	12	32
SiO <sub>2</sub>	10	1	11
B	5	1	6
TDS	1280	292	1572

## Boiler Fireside and Air Preheater Washing

Wash water from boiler fireside and air preheater cleaning will be a once per year input to the ash lagoons of approximately 7850 m<sup>3</sup> (average annual flow rate of 0.25 l/s) per unit. This water is generally of poor quality containing flyash and soot, rust, magnesium salts and metallic ions. Copper, iron, nickel and chromium are usually prevalent. Soda ash or other alkaline materials may be used to enhance cleaning and, hence, can appear in the washing fluid although there has been no indication that they will be used at Poplar River.

Flow wise, annual boiler fireside and air preheater wash water is equivalent to less than 0.2 percent of the ash slurry input to the ash lagoons, and approximately 1 percent of the discharge from the main boiler house sump. However, concentrations of some elements are likely to be much higher than that picked up in the ash slurry. Typical concentrations (U.S. EPA, 1974) of some elements in these types of wastes are as follows:

<u>Element</u>	<u>Concentration (mg/l)</u>
TDS	10,000
TSS	2,200
Na	500
Mg	1,000
SO <sub>4</sub>	1,200
C1	5
P	1.5
Fe	1,500
Cr	10
Ni	25
Cu	5
Zn	5

Since these are only typical values from operating power plants and, for most elements, are considerably higher than intake water concentrations, they can be considered as rough approximations of the concentration additions to be expected in the washwater.

#### Coal Gallery Drains Discharge

The estimated average flow of wastewater from washdown of the coal handling gallery drains is 1 l/s for 1 unit and 2 l/s for 2 units. In order to establish the amounts of various elements readily dissolvable from the coal, the Plant, Mine and Reservoir Operations Committee had leachate tests conducted by the University of Wisconsin on a Coronach area lignite sample using a 100,000 mg/l distilled water based leachate solution. The results of those tests are provided in Table 5-8.

It is probably reasonable to assume that the coal gallery washdown water will have coal concentrations similar to that used in the leachate tests (i.e. in the 100,000 mg/l range). The concentrations listed in Table 5-8 would then be representative of the increase in washwater concentration levels. Using assumed average reservoir water concentrations, the washdown water quality would be as follows:

<u>Element</u>	<u>Avg. Reservoir Water Conc. (mg/l)</u>	<u>Conc. Added From coal (mg/l)</u>	<u>Washdown Water Conc. (mg/l)</u>
Na	200	120	320
K	10	5	15
Ca	50	37	87
Mg	50	34	84
B	5	15	20
HCO <sub>3</sub>	450	(0)	(450)
SO <sub>4</sub>	500	440	940
NO <sub>3</sub>	10	(0)	(10)
C1	20	(5)	(25)
SiO <sub>2</sub>	10	(4)	(14)
TDS	1280	641	1921

The bracketed figures shown in this table are estimated from the ash leachate tests and do not reflect any actual coal leachate test results.

Although the washdown water flow is less than 1 percent of the ash slurry flow, the high Na concentration indicated suggests that Na addition from the washdown could be up to 3 percent of the quantity added to ash lagoon water from the ash. The addition of other elements is not sufficiently higher than ash leachate additions to make any significant difference.

TABLE 5-8        RESULTS OF COAL LEACHATE TESTS<sup>1</sup>

<u>Major and Minor Components</u>	<u>Mean Concentration (mg/l)</u>
Na	118.0
K	4.86
Mg	33.6
Ca	36.9
B	15.0
SO <sub>4</sub>	435.5
<u>Trace Components</u>	
Mn	0.187
Fe	0.477
Sr	0.706
V	0.016
Cr	0.018
Co	0.12
Ni	0.12
Cu	0.015
Zn	0.015
Cd	0.015
Al	0.231
Pb	0.12
As	0.12
Se	0.12
Zr	0.016
Sb	0.12

<sup>1</sup>These experiments were conducted by the University of Wisconsin, Water Chemistry Laboratory.

## Excess Water From Evaporation Pond

The evaporation pond has a capacity to hold a volume slightly less than that which would occur with an average rainfall year assuming all precipitation occurred as runoff from the coal storage area. It is estimated that at least half of the annual precipitation will either evaporate from the coal pile or be lost to seepage. These losses combined with direct evaporative loss from the pond are expected to make the need for releases to the ash lagoons infrequent and during only the higher rainfall years.

### (b) Water Quality in Ash Lagoons

In estimating the quality of water in the ash lagoons, all waste discharges into the lagoons must be considered. Chemical equilibrium conditions in the lagoons will influence the amount of initially dissolved material that will precipitate and the amount that will remain in solution. The equilibrium conditions may in turn be affected by the changing quality of the reservoir water being used to slurry ash, being treated for boiler make-up or being used for ash system cooling. When good water quality exists in the reservoir, higher percentages of some leached elements may remain in solution than when the reservoir water quality is poor. Certain elements, on the other hand, will never reach their solubility limits and hence will remain in solution. The solubility product criterion is often used for determining saturation levels of compounds. This criterion is based on the rule that at the limit of solubility or the brink of precipitation, the product of the concentrations (in moles/litre) of the component ions of a substance is constant at a given temperature. When there are a large number of diverse ions from diverse sources in diverse proportions, the solubility product is a much better indicator of the saturation level than mere solubility in mg/l for a pure substance. To properly analyze the chemical equilibrium conditions in the ash lagoons, an equilibrium model capable of computing saturation levels and precipitation rates over time for a complex set of interrelated processes and variable input concentrations would have to be used.

Table 5-9 summarizes the various waste inputs to the ash lagoons, their estimated quality and the resulting water quality in the lagoons that might be expected under conditions of elevated TDS in Cookson Reservoir, such as may occur during an extended dry period. The data presented is based on a simple mass balance adjustment to the total water volume in the lagoon. Considered in the mass balance adjustment is the estimated average net evaporation loss from the lagoons of 800,000 m<sup>3</sup>/year (based on an average 0.6 m annual evaporation loss from the combined surface area of three lagoons). The evaporation loss represents 7 percent of the total volume input to the ash lagoons per year. Seepage loss from the lagoons was not considered in the mass balance as seepage flow will be similar in quality to ash lagoon water.

The figures in Table 5-9 suggest that of the waste streams to be discharged into the ash lagoons, the ash slurry itself will contribute in

TABLE 5-9 INPUTS TO ASH LAGOONS AND TYPICAL LAGOON WATER QUALITY DURING AN EXTENDED DRY PERIOD

ELEMENT	QUANTITIES ADDED FROM WASTE STREAMS										TYPICAL CONCENTRATION IN ASH LAGOONS <sup>1</sup>		
	ASH SLURRY (262.2 l/s)	MAIN BOILERHOUSE SUMP (68.2 l/s)	WATER TREATMENT PLANT (11.8 l/s)	FIRESTOE AND PREHEATER WASHING (0.5 l/s)	COAL GALLERY WASHING (1.0 l/s)	ALL WASTE STREAMS (343.7 l/s)			AVERAGE INTAKE WATER CONCENTRATION	BEFORE EVAPORATION	AFTER EVAPORATION		
mg/l	kg/day	Percent of Total	mg/l	kg/day	Percent of Total	mg/l	kg/day	Percent of Total	mg/l	kg/day	Percent of Total	mg/l	mg/l
Na	30	680	88.9	0		52	53	6.9	500	22	2.9	120	200
K	1.0	23	94.3	0		1.0	1.0	4.1			5	0.4	24.4
Ca	0	0	0.0	0		1.0	1.0	23.8			37	3.2	11
Mg	0	0	0.0	0		1.0	1.0	2.1	1000	43	91.7	34	50
C1	5	113	90.0	0		12	12	9.5	5.0	0.2	5	0.4	5.6
												0.3	24
HCO <sub>3</sub>	-200	-4531	-101.6	0		72	73	1.6			0	0.0	450
CO <sub>3</sub>	40	906	100.0	0		0	0				0	0.0	300
SO <sub>4</sub>	180	4078 <sup>2</sup>	94.3	0		153	156	3.6	1200	1.2	440	38	31
NO <sub>3</sub>	0	0	0.0	0		1.0	1.0	100.0			0	0.0	0
SiO <sub>2</sub>	4	91	98.5	0		1.0	1.0	1.1			4	0.4	500
B	10	227	99.0	0		1.0	1.0	0.4	0.04	0.0	1500	15	10
Fe	0.5	11.3	14.8	0		0.04	0.04	0.0			65	0.5	10
												0.0	13
												0.4	14
												2.0	2.2

1. Concentrations listed are based on a materials balance only. Certain elements, such as Mg and Fe, are likely to precipitation in actual operation.

2. Sample computation:  $262.2 \text{ l/s} \times 180 \text{ mg/l} \times 10^{-6} \text{ kg/mg} \times 86,400 \text{ s/day} = 4078 \text{ kg/day}$ .

excess of 90 percent of the added concentration for many of the elements listed including boron. The boiler fireside and air preheater washwaters could add large portions of the Mg and Fe increase. However, the Ca, Mg and Fe added by the smaller wastestreams may in reality be partially or totally precipitated in the lagoons. Initial high Ca and Mg additions from the ash were found to precipitate out fairly quickly in leachate tests, this being the reason for the zero addition from the ash slurry indicated in the table. The  $\text{HCO}_3$  reduction of 200 mg/l is likely indicative of what would occur in actual practice with a high alkalinity intake water. However, under average conditions, the  $\text{HCO}_3$  reduction would probably not be greater than 100 mg/l.

(c) Verification of Ash Lagoon Water Quality Estimates

In order to verify the assumptions made in transforming ash leachate test results to ash lagoon water quality, an ash lagoon water quality study was conducted at the Boundary Dam generating station Unit #6. Unit #6 at Boundary dam, which has a 95 percent plus electrostatic precipitator, has been operating since November, 1977. The lagoon for Unit #6 was essentially empty prior to the commencement of operation. Performance tests indicated that during the study, the precipitator was operating at greater than 98 percent efficiency.

In the study, samples were taken twice during one of the daily 8 hour shifts at the plant for a 5-day period in August, 1978. The samples included:

- (i) intake slurry water (reservoir water)
- (ii) pulverized coal
- (iii) bottom ash (wet)
- (iv) flyash (dry)
- (v) bottom ash slurry
- (vi) flyash slurry
- (vii) lagoon water

In addition to the twice daily reservoir water samples taken for analysis, larger reservoir water samples were also taken twice daily and composited into one bulk sample for use in the ash leachate solution.

Each of the liquid samples taken was retained separately for individual analysis. Composite samples of pulverized coal, bottom ash and flyash were made up for the 5-day period from the individual twice daily samples.

Static leaching tests were run separately by Montana State University and the Environmental Protection Service, Environment Canada on the composited bottom ash and flyash samples, and on a combined ash obtained by mixing portions of the composited bottom ash and flyash in the ratio 58 percent flyash to 42 percent bottom ash. This ratio has been determined from actual operating experience at Boundary Dam. The ash to reservoir water ratio used in the leachate tests was 55,000 mg/l reflecting the average ratio in the ash slurries to the Unit #6 lagoon. The results of the leachate tests are summarized in Table 5-10, which also shows the average values obtained from analysis of the lagoon water samples. Detailed results of the tests are contained in Annex F.

TABLE 5-10 BOUNDARY DAM ASH LAGOON STUDY RESULTS

Parameter	Conc. in Reservoir Water	Leachate Concentrations						Mean Conc. in Unit #6 Lagoon (5 days)
		Total	Bottom Ash Added	Total	Flyash Added	Total	Composite Ash Added	
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
<b>Montana State University 31-day leachate tests</b>								
Na (mg/l)	92.2	113.6	+21.4	117.5	+25.3	114.5	+22.3	187.6
K (mg/l)	11.2	12.8	+1.6	14.0	+2.8	11.5	+0.3	11.7
Ca (mg/l)	44.4	31.9	-12.5	50.1	-5.7	56.4	+12.0	59.7
Mg (mg/l)	22.3	26.1	+3.8	0.6	-21.7	1.2	-21.1	20.4
HCO <sub>3</sub> (mg/l)	214.0	192.0	-22.0	36.8	-177.0	94.5	-120.0	178.0
CO <sub>3</sub> (mg/l)	-	10.0	-	55.6	-	36.9	-	10.8
SO <sub>4</sub> (mg/l)	210.0	235.0	+25.0	273.0	+6.3	240.0	+30.0	446.0
NO <sub>3</sub> (mg/l)	0.9	0.6	-0.3	2.6	+1.7	0.4	-0.5	0.8
C1 (mg/l)	10.3	10.4	+0.1	13.0	+2.7	11.6	+1.3	11.4
SiO <sub>2</sub> (mg/l)	4.7	3.2	-1.5	2.6	-2.1	7.4	+2.7	4.2
B (mg/l)	0.3	2.5	+2.2	6.3	+6.0	6.1	+5.8	2.7
<b>Environment Canada 21-day leachate tests (mean from 3 labs)</b>								
Na (mg/l)	82.0	124.0	+42.0	136.0	+54.0	125.0	+43.0	
K (mg/l)	11.6	12.1	+0.5	13.9	+2.3	13.3	+1.7	
Ca (mg/l)	42.9	39.8	-3.1	62.8	+19.9	75.7	+32.3	
Mg (mg/l)	20.9	19.0	-1.9	0.7	-20.2	1.4	-19.5	
HCO <sub>3</sub> (mg/l)	187.0	188.0	+1.0	93.3	-93.7	237.0	+50.0	
CO <sub>3</sub> (mg/l)								
SO <sub>4</sub> (mg/l)	202.0	295.0	+93.0	311.0	+109.0	336.0	+134.0	
NO <sub>3</sub> (mg/l)	3.5	2.8	-0.7	4.2	+0.7	3.3	-0.2	
C1 (mg/l)	25.6	17.7	-7.9	23.4	-2.2	13.0	-12.6	
SiO <sub>2</sub> (mg/l)	2.3	6.8	+4.5	1.0	-1.3	0.8	-1.5	
B (mg/l)	0.3	3.6	+3.3	14.6	+14.3	14.6	+14.3	

The data provided in Table 5-10 shows that considerable variability exists between the results for different ash samples and that caution should be used in extrapolating from a small number of leachate tests to actual lagoon conditions over an extended time period. If the ash were consistent, reservoir water quality fairly constant and the leachate tests reflective of dissolution rates, one would expect good agreement between columns (7) and (9) in Table 5-10. Although there is good agreement for some elements, there is considerable difference for others. One of the largest percent differences is between the Montana State University and Environment Canada dissolution rates for B for the flyash and composite ash samples. There is good agreement on dissolution from the bottom ash.

It is not known whether the large differences in the measured dissolution rates for B are due to differences in the sample splits tested by the different laboratories or to reliability of the test methods used. Since there was good agreement by all laboratories on B in the reservoir water and the bottom ash leachate, there is reason to suspect a variability in the flyash samples. However, with the sample preparation methods used, good consistency would have been expected. All laboratories used the curcumin method for B analyses, which can be affected by a high solids level in the solution. Good agreement was obtained on all B analyses conducted by the three laboratories contracted by Environment Canada suggesting the high B readings obtained by those laboratories are reliable. Based on the one preliminary analyses received showing B concentration in the flyash and bottom ash of 550 ug/g and 290 ug/g respectively, up to 30 mg/l of B could theoretically dissolve into the leachate solution from the flyash and 16 mg/l from the bottom ash. The 15 mg/l measured is about 50 percent of the B available in the flyash. The composite ash has a lower B concentration than the flyash but still picks up about 15 mg/l in leaching. It would appear from the various leaching experiments conducted that B will leach from flyash up to about 15 mg/l within the limits of B availability. Even at surplus B availability, few concentrations above the 15-17 mg/l range were observed in any leaching experiments conducted to date.

To use the Boundary Dam study results in assessing the validity of directly extrapolating leachate test results to lagoon water quality, the leaching results must be compared with conditions in the Unit #6 lagoon. The average B concentration in the Unit #6 ash lagoon during the 5-day sampling period was 2.7 mg/l and levels ranging from 2.0-5.1 mg/l were recorded during the summer of 1978. Average B levels in the reservoir water used for ash transport during the time period required for the last full turnover of the lagoon is estimated to be about 3.0 mg/l based on the results of periodic reservoir water sampling during that period, and the average reservoir level during the 5-day sampling period was 0.37 mg/l. This suggests that under actual operation, dissolution of B from the ash probably adds on average from 1 to 5 mg/l to the ash lagoon water. The 5.1 mg/l recorded level was probably achieved through a short time period

during which B dissolution was greater than 5 mg/l. The average dissolution during operation of 1 to 5 mg/l is considerably less than the 5.8 to 14.3 mg/l range obtained in the leachate tests. Hence, it must be concluded that the leachate tests tend to overestimate dissolution of B from flyash. There has, however, been a recent increase in the Unit #6 ash lagoon B concentration as indicated by the 5.1 mg/l value in November, 1978. This may be linked to the recently determined precipitator operating efficiency of 98 percent plus and could suggest a trend towards increased B levels in the lagoon. With the 99.5 percent precipitator at Poplar River a much higher percentage of the fine flyash particles will be collected. Although B dissolution is not likely to be as high as the maximum levels recorded in the Poplar River ash leachate tests, it could be higher than the levels observed in operation of Unit #6 at Boundary Dam. The "representative value" of 10 mg/l listed in Table 5-7 is lower than the leachate test value of 15.5 mg/l and should be considered the current best estimate of average B addition at Poplar River.

Comparisons between leaching test results and actual operation can be made for other constituents using the data provided in Table 5-10 and Annex F. Although the magnitude of the changes occurring when the combined ash is added to reservoir water (column 7) do not always agree with the difference between reservoir and lagoon water (column 9) during the 5-day test period, the trends are similar. The increase in SO<sub>4</sub> and decrease in HC0<sub>3</sub> is consistent with the observed reactions in the lagoon, although a good correlation on magnitudes of these changes is not indicated. An increase in Na uptake, with Na remaining in solution as Na<sub>2</sub>SO<sub>4</sub>, and the precipitation of Mg as Mg(OH)<sub>2</sub> are consistent with conditions of actual operation.

(d) Discharge from Ash Lagoons

As a result of the initial ash leachate test results on Poplar River ash and the verification study on the Boundary Dam Unit #6 ash lagoon, it is concluded that the figures contained in Table 5-7 are reasonable approximations of expected dissolution of elements from the ash at Poplar River and the concentrations in Table 5-9 are indicative of the effluent quality that might be expected from the lagoons with a high alkalinity intake water. However, as discussed previously, considerable variation in ash lagoon water concentrations can be expected. The HC0<sub>3</sub> reduction is likely to be less on average than the 200 mg/l reduction indicated in Table 5-9. Reductions of 10 to 40 mg/l were observed between the reservoir and lagoon at Boundary Dam during the test period. A 100 mg/l reduction at Poplar River would be a more realistic assumption for average conditions. Both the B and SO<sub>4</sub> additions to ash slurry water at Poplar River are expected to be somewhat higher than those being observed at Boundary Dam and the values in Table 5-7 should be reflective of average operating conditions.

The quality of effluent from the Poplar River station ash lagoons is dependent not only on the dissolution of material from the ash, but on other factors such as intake water quality and concentration due to evaporation. Hence, the effluent quality will vary over time as these factors vary. The major controlling factors have been incorporated into the reservoir operation model which can be used to compute ash lagoon effluent

quality as well as reservoir and border water quality on a monthly basis over an extended time period. Some results obtained from model runs made to evaluate mitigation requirements are contained in Section 8.

### 5.3 Inputs and Losses to Groundwater System

#### 5.3.1 Sanitary Waste Disposal

##### (a) Construction Camp

The camp's water supply is obtained from wells near the camp. Using an average camp population of 300 and a flow of 295 l (65 gallons) per capita per day, it could be assumed that the following amounts of nutrients have been lost into the soil beneath the temporary sewage lagoon (in addition to the inorganic constituents in the ground water):

- BOD <sub>5</sub>	6.6 kg/day
- Phosphate	1.32 kg/day
- Nitrogen	2.66 kg/day
- MBAS	0.089 kg/day

Over the short life of this lagoon system, the added constituents might never reach the ground water, but it would be more conservative to assume that the ground water could be affected.

##### (b) Power Plant

Seepage losses to groundwater from the sewage irrigation area have not yet been determined. A properly designed and operated sewage treatment irrigation scheme should have no significant effect on the groundwater quality or on the water quality in Cookson Reservoir. However, the uncertainty about the subsurface conditions and the proximity of the adjacent drainage ditch, which may provide a discharge avenue for seepage water, suggests that the area be investigated further. It may well be that seepage to the groundwater system would be quickly released again to the ditch.

The conditions required by Saskatchewan Environment for construction and operation of both the sewage lagoon and the irrigation area should be sufficient to ensure that no significant inputs to the groundwater system occur.

#### 5.3.2 Coal Storage Area

The evaporation pond collects runoff from the coal pile. The coal pile area runoff, estimated to average 1.1 l/s for either 1 or 2 units, can be expected to have chemical concentrations similar to those listed in Section 5.2.6 for the coal leachate tests. Since it is not known whether the evaporation pond area is to be lined, or what the exact nature of subsurface material beneath the pond is, no firm estimates of seepage losses to groundwater have been made.

Seepage from the coal pile itself is estimated to be about 1/4 of the average annual precipitation of 0.355 m. This gives an average seepage rate of 13,310 m<sup>3</sup>/year or 0.42 l/s.

### 5.3.3 Seepage from Ash Lagoons

#### Seepage Rates

The report on the ash lagoon area geotechnical investigation (Ground Engineering, 1978) was the primary source of technical information used by the Plant, Mine and Reservoir Operations Committee in making seepage loss predictions for the Poplar River ash lagoons. The report suggests that: "The general direction of groundwater movement will be downwards towards the underlying gravels which will act as a drain. However, there will be substantial movement of seepage through the till leading to saturation of the ground surface immediately beyond the perimeter of the confining dikes."

The downward seepage rate will depend upon the permeability of the upper till layer and its consistency over the lagoon area as well as the carrying capacity of the underlying Empress Group sands and gravels. Ash lagoon water that does seep down to the Empress Group layer is likely to be conveyed horizontally along that layer in a southeasterly direction towards the East Poplar River below Morrison Dam. The location of an assumed seepage path is shown on Figure 5-1 and a subsurface profile along this path is shown on Figure 5-12. The description in Section 5.1.13 (c) suggests that there is reason to believe that the upper till layer may not be consistant and that localized areas of higher permeability sands and gravels are a distinct possibility. Because of this, several different estimates of downward seepage loss were made. These were:

- (i) for a consistant till layer with a coefficient of permeability (K) of  $10^{-6}$  cm/s,
- (ii) a more pervious but consistant till layer with a K of  $10^{-5}$  cm/s
- (iii) the major portion of the till layer having a K of  $10^{-6}$  cm/s but isolated pockets of sand and gravel totalling 10,000 m<sup>2</sup> in area with a K of  $10^{-3}$  cm/s

The seepage computations for the three conditions yielded the following downward seepage rates for lagoons 1, 2 and 3 (1,379,000 m<sup>2</sup>) :

<u>Condition</u>	<u>Seepage Rate l/s (cfs)</u>	
(i)	6.13	(0.22)
(ii)	61.29	(2.16)
(iii)	50.69	(1.79)

Computations were made using the Darcy's Law equation:  $Q = AK \frac{dh}{dl}$

where  $Q$  = rate of seepage flow ( $m^3/s$ )

$A$  = the cross sectional area of the seepage flow path ( $m^2$ )

$K$  = coefficient of permeability ( $m/s$ )

$dh/dl$  = hydraulic gradient ( $m/m$ )

Additional computations were carried out to determine whether the Empress Group sand and gravel layer would have sufficient carrying capacity to convey these seepage flows and in effect act as a drain, or whether the conveyance capacity of that layer would be limiting. The ash lagoon geology report (Ground Engineering, 1978) lists Empress Group permeability coefficients ( $K$ ) ranging from  $4.0 \times 10^{-5}$  to  $6.0 \times 10^{-3} \text{ cm/s}$ . It also states, however, that one hole had to be abandoned because "the gravels were so coarse it became impractical to install a piezometer and develop it properly." The permeability coefficient at that location can be expected to be greater than the range indicated above. The seepage rate computations for the Empress Group show that with a seepage path width through the layer of twice the average width of lagoons 1, 2 and 3 and a  $K$  of  $1 \times 10^{-3} \text{ cm/s}$ , the carrying capacity of the layer would be only about  $0.44 \text{ l/s (.016 cfs)}$ . An average  $K$  of  $1.4 \times 10^{-2} \text{ cm/s}$  is needed before the lowest estimated downward seepage rate of  $6.13 \text{ l/s}$  is equalled. This suggests that the Empress Group capacity could be a limiting factor and the layer may not act as a drain. However, if the flow through the layer from the ash lagoons is less unidirectional and more radial, there would be adequate capacity at a  $K$  of  $1 \times 10^{-3} \text{ cm/s}$ .

The Ground Engineering report suggests that there could also be substantial radial seepage from the ash lagoons with horizontal water movement through the upper till. This could lead to direct discharge of seepage flow to the cooling water discharge channel and possibly some movement westward towards the mine area if the groundwater cone of depression extends to the vicinity of the ash lagoons. Most of the seepage to the cooling water channel is likely to be beneath as opposed to through the containment dikes. Analysis shows that with the impervious core and blanket proposed for the dikes, seepage through the dikes is likely to be less than  $0.16 \text{ l/s (0.0056 cfs)}$ . However, flow through the blanket and fractured upper till beneath the dikes towards the cooling water channel is estimated to average  $2.1 \text{ l/s (0.074 cfs)}$  for three lagoons and a  $K$  of  $1 \times 10^{-5} \text{ cm/s}$ . Total seepage radially in all directions from three lagoons is computed to be about  $5.4 \text{ l/s (0.19 cfs)}$ .

Because of the potential effect of seepage on both ground and surface water quality, an independent study of ash lagoon seepage potential (Geraghty and Miller, 1979) was contracted by the Plant, Mine and Reservoir Operations Committee. The report on this study is contained in Annex G.

Seven different scenarios were examined in the study with different combinations of ash lagoon numbers and locations reflecting potential once-through and recirculating ash disposal alternatives. The draft report concludes that seepage from the ash lagoons will move essentially vertically downward to the Empress Group through the till and move south/southeast to discharge into the East Poplar River between Morrison Dam and the Internatioanl Boundary. It further concludes that seepage laterally through fractures in the till to the cooling water discharge channel and small surface ponds and marshes should be minimal. Although a wide range of potential seepage rates are presented, the report suggests that seepage downward through the till is likely to be in the 1.2 to 12 l/s range for three lagoons operating in series if the upper till layer is rolled as proposed and a buildup of ash has occurred. Prior to additional sealing by settled ash, seepage could be higher. The report states that Empress Group conveyance capacity is not expected to be a controlling factor. Whatever seepage gets through the lining and till layer will be conveyed by the Empress Group with larger seepage rates simply creating a more extensive seepage plume and a longer discharge reach along the East Poplar River.

The Geraghty and Miller conclusions on principal seepage flow direction and range of potential seepage rates are similar to those of the Plant, Mine and Reservoir Operations Committee. However, it is felt that seepage back to Cookson Reservoir should not be assumed to be negligible. The reservoir water level will vary during operation giving rise to a fluctuating hydraulic gradient between the ash lagoons and the reservoir. Although it may not occur under all operating conditions, the groundwater mound created by the ash lagoons may give rise to localized flow reversal back to the reservoir under some conditions. It is recommended that a rate of 5.0 l/s be considered in the reservoir operation model analyses for the recirculating combined ash lagoon alternative. Seepage to the reservoir can be ignored in assessing the proposed system since it is negligible in comparison to the decant.

Based on the foregoing, the following rates of seepage from the ash lagoons to the East Poplar River have been selected for use in the reservoir operation model study of flow distribution at the International Boundary:

Low rate	0.5 l/s (0.018 cfs) per lagoon
Average rate	2.0 l/s (0.07 cfs) per lagoon
High rate	17.0 l/s (0.6 cfs) per lagoon

The proposed method of lining the lagoons by excavating and recompacting the surface layer of till is expected to ensure a maximum permeability of  $1.5 \times 10^{-5}$  cm/s. This is within the range of K values used above in calculating seepage loss and would not substantially alter the seepage estimates. Once ash has built up in the lagoons, some degree of self-sealing may occur due to the fine grained consistency of the ash and reduced permeability could result.

### Seepage Quality

To determine whether seepage water quality is likely to be different from ash lagoon water quality, column leachate tests were conducted on a Poplar River ash as described in Annex E. The results of the tests are listed in Table 5-11 along with the selected "representative" additions to ash lagoon water quality from Table 5-7. The tests were run with both reservoir water and concentrated reservoir water as leaching solutions with flow rates of 0.43 ml/hr and 0.37 ml/hr respectively. With a 1 cm diameter column, this is a downward seepage velocity of  $1.5 \times 10^{-4}$  cm/s which is much higher than the expected downward seepage through the base of the lagoons of about  $1 \times 10^{-6}$  cm/s. A danger in small sample column tests of this type is that piping or channelization through the sample can occur leading to erroneous results. It is not known whether piping occurred in these tests.

It is difficult to draw any firm conclusions from the data in Table 5-11. For some elements, the fraction of leachate water collected after 48 hours showed a significantly increased concentration whereas the fraction collected after 150 hours showed concentrations lower than that of the reservoir water used as the leaching solution. The 150 hour results showed decreases in Ca, Mg, Na, SO<sub>4</sub> and HCO<sub>3</sub> from leaching solution levels with a decrease in specific conductance of 280 umhos/cm. The apparent precipitation of Na<sub>2</sub>SO<sub>4</sub> and Ca and Mg as hydroxide or carbonate salts is attributed to the high pH level maintained by a lack of opportunity to equilibrate with CO<sub>2</sub>. Whether that amount of precipitation would occur in actual practice is uncertain. It is more conservative to assume that seepage flows would have a quality similar to that observed in the static leachate tests. It is of interest to note that boron levels increased with time, the maximum addition being about 4.5 mg/l. This could result from increased solubility with the slight drop in pH from about 11.7 to 10.0.

#### 5.3.4 Seepage From Holding Pond

Seepage from the holding pond is expected to be extremely low because of the low head differential and small area of the pond. Experience to date has shown that the pond has effectively retained the local runoff water with which it has been filled since the spring of 1978.

The holding pond will, however, contain very highly contaminated wastewater having, at time, extreme pH values. Even small seepage flows would be undesirable and warrant precautionary measures.

TABLE 5-11 COLUMN LEACHATE TEST RESULTS - POPLAR RIVER ASH

ELEMENT	CONCENTRATION (mg/l) ADDED IN COLUMN LEACHING				REPRESENTATIVE VALUE	STATIC LEACHING REPRESENTATIVE VALUE (mg/l)		
	RESERVOIR WATER		CONC. RESERVOIR WATER	18.1 ml 48 hours				
	20.3 ml 48 hours	73.6 ml 171 hours						
Na	358	-22	301	-41	0	30		
K	27	0	21	-2	0	1		
Ca	-19	-34	-10	-13	-15	0		
Mg	-33	-33	-59	-59	-40	0		
B	2.3	4.5	0.7	4.4	4	10		
HCO <sub>3</sub> <sup>-</sup>	-	-268	-	-269	-270	-200		
CO <sub>3</sub> <sup>=</sup>	188	117	-21	168	120	40		
SO <sub>4</sub> <sup>=</sup>	429	-147	389	-293	-100	180		
NO <sub>3</sub> <sup>-</sup>	10	4	8	3	5	0		
C1 <sup>-</sup>	28	3	7	5	5	5		
SiO <sub>2</sub>	10	14	-18	13	10	4		
Cr	.10	.10	.10	.10	.10	.10		
Cd	.02	.02	.02	.02	.02	.02		
Cu	.04	.04	.04	.04	.04	.10		
Fe	.35	.35	.35	.35	.35	.35		
Mn	.025	.025	.025	.025	.025	.02		
Ni						.06		
Pb	.005	.005	.005	.005	.005	.01		
Zn	.02	.02	.02	.02	.02	.01		
Al	2.0	2.0	2.0	2.0	2.0	15		

## 5.4 Deposition from Atmospheric Emissions

### 5.4.1 General

Emissions from the Poplar River power plant stack, and fugitive dust emissions from both the plant and the mining activities, will result in the deposition of emitted material on surrounding land and water surfaces. For use in assessing the potential impact of these emissions on the water resources of the area, estimates were obtained for emission rates, resulting ground level concentrations and deposition quantity.

This information was obtained from separate atmospheric emission studies by the U.S. Environmental Protection Agency (Thoem, 1978) and the Atmospheric Environment Service (Portelli, 1975), Environment Canada. Although somewhat different assumptions and methodologies were used by the two agencies, the results are sufficiently definitive to permit a rough determination of the deposition quantities that can be expected.

### 5.4.2 Deposition from Stack Emissions

#### (a) U.S. Environmental Protection Agency (EPA)

Parameters examined in the EPA stack emission study include total suspended particulates (TSP), sulphur dioxide ( $\text{SO}_2$ ) and nitrous oxides ( $\text{NO}_x$ ). The following factors and assumptions were utilized in the analysis:

##### (i) Stack Data

Height	122.0	m
Diameter	7.11	m
Exit Velocity	24.4	m
Exit Temperature	424°	K

##### (ii) Control Efficiency

- for TSP	99.5%
- for $\text{SO}_2$ & $\text{NO}_x$	0%

##### (iii) Emission Rates

	<u>kg/<math>10^6</math> BTU</u>	<u>g/s</u>	
		<u>300 MW</u>	<u>600 MW</u>
TSP	.036	28	56
$\text{SO}_2$	.880	676	1352
$\text{NO}_x$	.272	227	454

The  $\text{SO}_2$  figures are based on a sulphur retention in the ash of 8%.

Computation of ground level concentrations were made using the Valley Model (Burt, 1977) with input wind data from the Glasgow, Montana meteorological station for the years 1967-71.

Figure 5-15 showing the mean annual concentrations of  $\text{SO}_2$  for the 600 MW case exemplifies the results obtained from the model. As can be seen on the figure, the maximum annual concentration of  $\text{SO}_2$  is about 1.2  $\mu\text{g}/\text{m}^3$  occurring 17-32 km northwest of the station. Concentrations in excess of 0.8  $\mu\text{g}/\text{m}^3$  occurred at a distance of 8-24 km southeast of the station.

Similar data was computed for TSP and  $\text{NO}_x$ . The maximum concentrations estimated for each parameter for the 300 and 600 MW cases are:

	<u>300 MW</u>	<u>600 MW</u>
TSP ( $\mu\text{g}/\text{m}^3$ )	0.025	0.05
$\text{SO}_2$ ( $\mu\text{g}/\text{m}^3$ )	0.6	1.2
$\text{NO}_x$ ( $\mu\text{g}/\text{m}^3$ )	0.2	0.4

The 300 MW values were approximated as being half the 600 MW values. This assumption is probably not valid but is sufficiently accurate for the analysis here.

Using the above maximum concentrations, and depositional velocities for TSP,  $\text{SO}_2$  and  $\text{NO}_x$  of 0.5 cm/s, 2.0 cm/s and 2.0 cm/s respectively, deposition rates were calculated. These are:

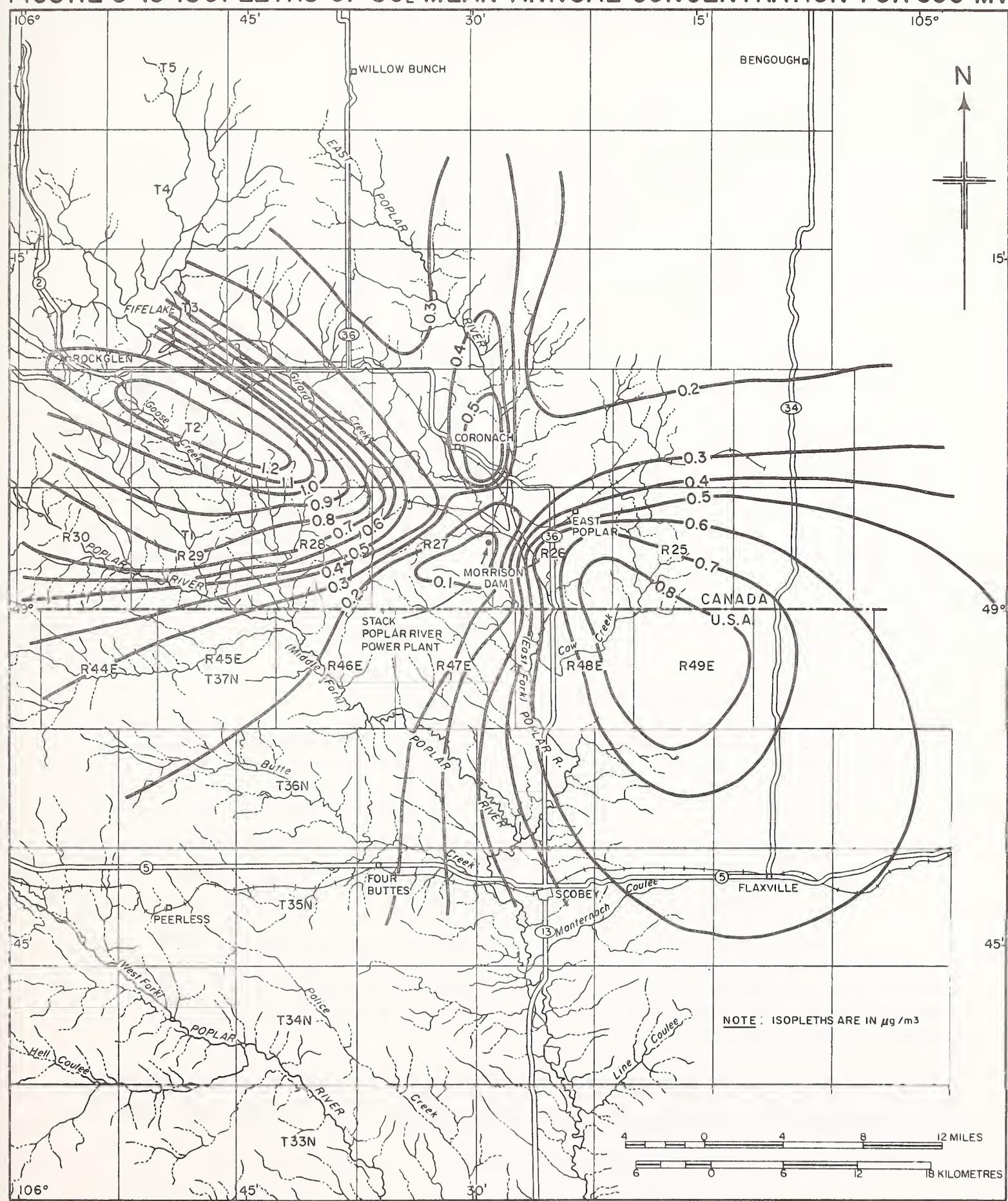
	<u>300 MW</u>	<u>600 MW</u>
TSP ( $\text{g}/\text{m}^2/\text{year}$ )	.016	.032
$\text{SO}_2$ ( $\text{g}/\text{m}^2/\text{year}$ )	.380	.760
$\text{NO}_x$ ( $\text{g}/\text{m}^2/\text{year}$ )	.072	.145

As a rough check on these rates, average annual deposition was computed using the emission rates provided above and assuming all emitted material would be deposited within a 70 km radius of the plant. With an average 80% plant capacity factor, these computations yield the following average deposition rates within that area for a 600 MW plant:

TSP	0.092 $\text{g}/\text{m}^2/\text{year}$
$\text{SO}_2$	2.217 $\text{g}/\text{m}^2/\text{year}$
$\text{NO}_x$	0.744 $\text{g}/\text{m}^2/\text{year}$

It can be seen from these computations that if all emitted material is assumed to be deposited within 70 km, the deposition rate would be about 3 times the annual "maximum" EPA estimates for TSP and  $\text{SO}_2$  and about 5 times the EPA  $\text{NO}_x$  estimates. A considerable portion of the fine particulates emitted must therefore be deposited beyond 70 km. It is not unrealistic to expect this, especially during the more stable winter air mass conditions (Portelli, 1978). Studies (Barrie and Whelpdale, 1978) for the Great Canadian Oil Sands plant have shown very small percentages of emitted  $\text{SO}_2$  and  $\text{NO}_x$  appearing as their respective compounds in snow packs in the vicinity of the station suggesting that these substances will

FIGURE 5-15 ISOPLETHS OF SO<sub>2</sub> MEAN ANNUAL CONCENTRATION FOR 600 MW



be transported even farther than the particules and more widely distributed, particularly during winter months. Hence, the EPA estimates of maximum annual concentrations within 30 km of the Poplar River station would appear to be realistic.

(b) Environment Canada

Other than the SO<sub>2</sub> emission rates, most of the basic data used by the AES in their dispersion calculations (Portelli, 1975) are the same as those used by the EPA. The AES assumed a rate of sulphur retention in the ash of 40% as opposed to 8% assumed by the EPA. (The Saskatchewan Power Corporation contends retention will be approximately 36% whereas the EPA suggests it cannot be assumed to be greater than 10 to 15%). Based on 40% retention, the AES computed the following SO<sub>2</sub> emission rates:

<u>kg/10<sup>6</sup> BTU</u>	<u>300 MW</u>	<u>600 MW</u>
<u>g/s</u>		
0.454	378	756

The AES made dispersion calculations using three different models: The Coning Model, the Inversion Breakup Model and the Limited Mixing Model. Twenty-four-hour and 1-hour maximum concentrations were computed but no estimates were made for annual maximum concentrations of TSP, SO<sub>2</sub> or NO<sub>x</sub>. By comparing the AES and EPA 24-hour and 1-hour values determined with various mixing heights and stability conditions, it would appear that there is generally reasonable agreement between the results obtained by the two agencies. It is therefore concluded that AES estimates of annual maximum concentrations would not have been significantly different from those of the EPA.

#### 5.4.3 Deposition From Fugitive Dust Emissions

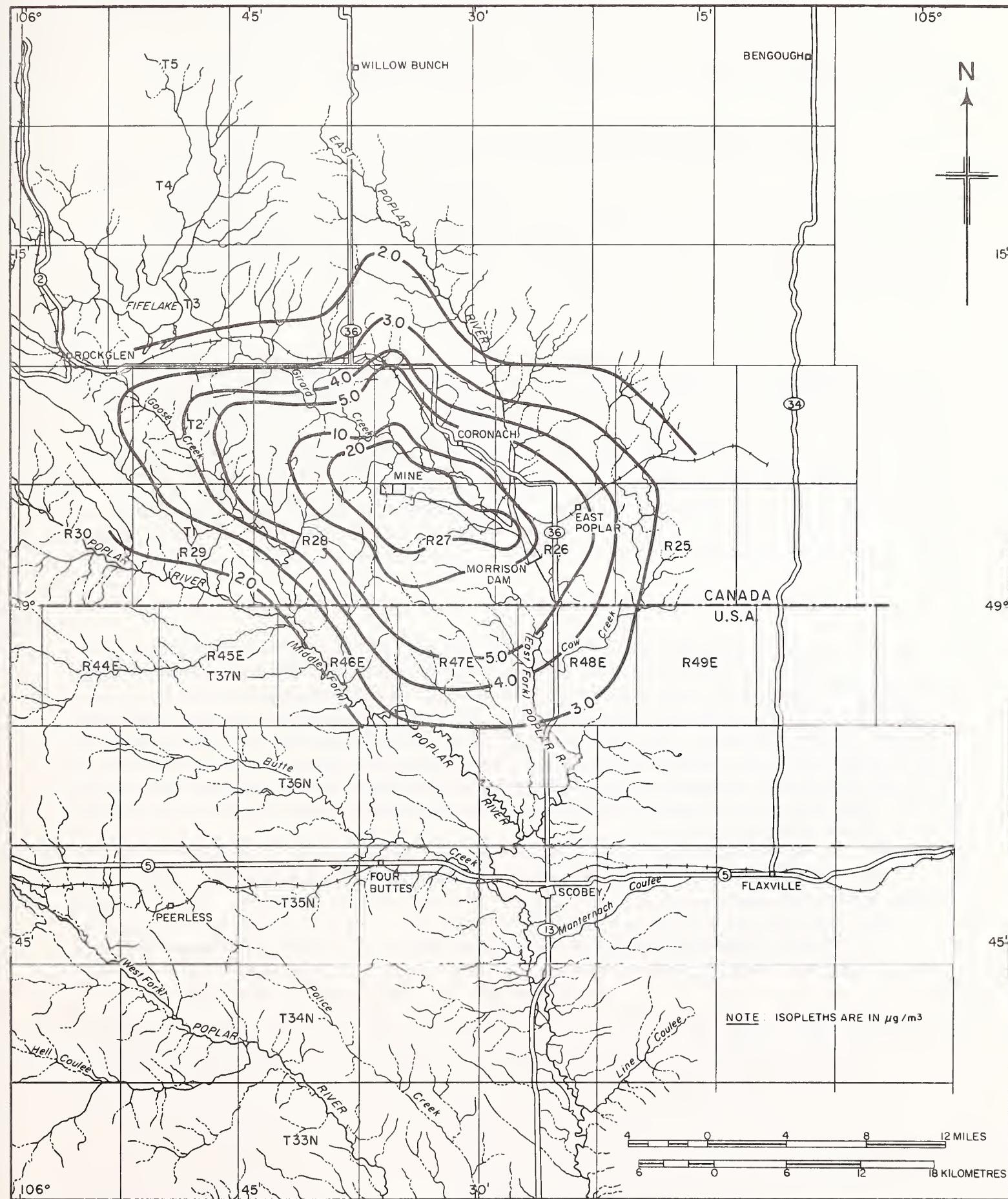
Annual emission of fugitive dust from the mining operation and activities around the Poplar River power plant are estimated to be:

	<u>Contributing Area (m<sup>2</sup>)</u>	<u>Quantity Emitted (kg/year)</u>
Mine	$2 \times 10^6$	$4.09 \times 10^6$
Plant	$0.12 \times 10^6$	$0.57 \times 10^6$

Using these emission rates, the EPA applied the Valley Model with the flat terrain option and annual STAR meteorological data for Glasgow, Montana for the years 1967-71. It was assumed that the total emissions consist of particles small enough to behave like a gas. The height of emissions was fixed at 10 m.

The computations carried out with the Valley Model produced the annual average particulate estimates shown in Figure 5-16. The fugitive dust concentration isopleths in the figure show a west-northwest to east-southeast axis of maximum annual average concentrations.

**FIGURE 5-16 ISOPLETHS OF MEAN ANNUAL PARTICULATE CONCENTRATIONS FOR 600 MW**



The concentrations over Cookson Reservoir would average 5 to 10 ug/m<sup>3</sup>. It should be noted that 10 ug/m<sup>3</sup> is 200 times the maximum annual TSP concentration estimated by the EPA for stack emissions implying that fugitive dust will be the major contributor to deposition on the reservoir. If the deposition velocity for suspended particulates is 0.5 cm/s, the fugitive dust deposition rate for a 10 ug/m<sup>3</sup> concentration is 6.3 g/m<sup>2</sup>/year.

#### 5.4.4 Effect on Surface Waters

The Plant, Mine and Reservoir Operations Committee has for the most part limited its treatment of atmospheric emissions to providing estimates of annual deposition rates. However, a few rough order of magnitude computations have been made to evaluate the significance of deposition inputs to surface waters.

##### (a) Particulate Deposition on Reservoir

Since TSP concentrations from stack emissions is estimated to be only a very small fraction of the fugitive dust concentrations, the deposition rate of 6.3 g/m<sup>2</sup>/year is reflective of the annual maximum deposition on the reservoir surface from both fugitive dust and stack emissions. 6.3 g/m<sup>2</sup>/year, when applied to the reservoir surface area of 7,368,620 m<sup>2</sup>, gives an annual deposition of 46,422,000 g/year on the reservoir. When related to the reservoir volume of 41,166,140 m<sup>3</sup>, this amounts to only about 1 mg/l increase in total solids in the reservoir per year and is considered insignificant.

##### (b) Boron Deposition on Reservoir

A rough estimate was also made for direct boron contributions to Cookson Reservoir from stack and coal related fugitive dust emissions. Coal related fugitive dust was assumed to be 20% of the total deposition estimated for the mine and power plant. Boron concentration in the coal was assumed to be 125 ppm. The boron collection efficiency for the 99.5% efficient electrostatic precipitator at Poplar River was assumed to be 94.12%. This is the boron removal efficiency determined in a study (Radian Corp., 1975) done for the Colorado Public Service Company's Comanche Station which has a 99.3% precipitator.

The plant TSP and fugitive dust deposition rates used for the boron estimates are:

Plant (based on EPA estimate, 600 MW)	.032 g/m <sup>2</sup> /yr.
Mine (20% of 6.3/m <sup>2</sup> /year)	1.26 g/m <sup>2</sup> /yr

Since the 600 MW plant operating at an 80% annual capacity factor is expected to consume  $3,489 \times 10^6$  kg of coal per year, the boron emitted will be:

$$3,489 \times 10^6 \times (125 \times 10^{-6}) (1.0 - .9412) = 25,640 \text{ kg/year}$$

The plant is expected to emit particulates at a maximum rate of 56 g/s which when reduced to an 80% annual capacity factor becomes 44.8 g/s average over the year or  $1.413 \times 10^6$  kg/year. Hence, boron concentration in the TSP is:

$$(25,640 / 1.413) = 18,150 \text{ ppm}$$

Boron deposition on the reservoir from the stack emission is therefore:

$$(.032 \text{ g/m}^2/\text{year}) (7,368,620 \text{ m}^2) (.01815 \text{ g/g}) = 4,280 \text{ g/year}$$

If all deposited boron was dissolved this would amount to an increase in dissolved boron in the reservoir of 0.00010 mg/l, which is insignificant.

The boron deposition on the reservoir from the fugitive dust emissions would be:

$$(1.26 \text{ g/m}^2/\text{year}) (7,368,620 \text{ m}^2) (125 \times 10^{-6}) = 1,160 \text{ g/year}$$

which is also a negligible addition.

(c) Selenium Deposition

Measurements of Se in the flue gas of the Comanche Station (Radian Corp., 1975) suggests an Se collection efficiency for the plants electrostatic precipitator of approximately 85 percent. However, a good material balance was not obtained with only about 23 percent of the total Se in the coal accounted for in the precipitator ash and sluice ash. Using a conservative assumption that up to 80 percent of the 0.8 ppm Se in the lignite at Poplar River could be emitted, the amount emitted annually by 2 units operating at an average annual capacity of 80 percent would be:

$$3,489 \times 10^6 \times (0.8 \times 10^{-6})(0.80) = 2,230 \text{ kg/year} = 0.071$$

This is 0.005 percent of the maximum annual SO<sub>2</sub> emission rate of 1,352 g/s listed in Section 5.4.2(a). If the deposition rates for Se and SO<sub>2</sub> are approximately the same, Se would be deposited at a rate of:

$$0.760 \times 10^6 \times 0.00005 = 38 \text{ ug/m}^2/\text{year}$$

A portion of the Se deposited would be on the winter snow and would be lost with the spring melt. As a means of comparison, if the entire amount

deposited was assumed to be mixed with the upper 20 cm of soil in the region of maximum deposition, it would contribute a concentration of:

$$(38 \times 10^{-3} \text{ mg/m}^2/\text{yr}) / (0.2 \text{ m}^3/\text{m}^2 \times 1600 \text{ kg/m}^3) = 0.00012 \text{ ppm}$$

Although no information was available on natural Se concentrations in the soil, it is expected that this addition would be only a very small fraction of the Se occurring naturally and, hence, insignificant.

6.1 Description6.1.1 Location and Physical Characteristics

The general location of the Cookson Reservoir is shown in Figures 2-1 and 5-1. Morrison Dam, on the East Poplar River approximately 3.5 km north of the International Boundary, creates a Y-shaped reservoir within the East Poplar River and Girard Creek valleys.

Shown in Figure 6-1 and Photo 6-1, Morrison Dam is a zoned earthfill embankment with a crest elevation of 757.0 m and an impervious blanket upstream to reduce seepage losses from the reservoir. The dam contains three outlet facilities to pass water downstream. These are a low level riparian outlet for scheduled low-flow releases, a gated service spillway for passing flood flows and an emergency earthfill plug spillway to supplement the service spillway in the event of an extreme flood. The riparian outlet and service spillway are located in the right (west) abutment and the emergency spillway at the end of the right wing dike to Morrison Dam.

At its full supply level (FSL) of 753.0 m, Cookson Reservoir has a length of approximately 11 km. Proceeding northward along the reservoir, the surface width narrows from about 1000 m at Morrison Dam to about 500 m some 2 km upstream of the dam and then expands again to about 1000 m at the Girard Creek junction. Throughout this 3 km core section, the depth of the reservoir remains relatively constant at 13 to 14 m. Causeways for road and railway crossings separate a small portion of the Girard Creek arm and about 2.5 km of the upper East Poplar River arm from the main body of the reservoir. The causeways contain culverts to permit water movement between the isolated sections and the main water body and the passage of flood flows downstream through the reservoir.

At its FSL of 753.0 m, the reservoir has a surface area of 7,368,620 m<sup>2</sup> and a volume of 41,166,140 m<sup>3</sup>. The surface area upstream of the causeway on the East Poplar River arm is about 1,150,000 m<sup>2</sup> or approximately 16 percent of the total surface area. Elevation-area and elevation-volume information for the reservoir is provided in Figure 6-2.

6.1.2 Method of Operation(a) Operating Policy

The reservoir has been created for the primary purpose of cooling condenser discharge water from the Poplar River generating station.

FIGURE 6-1 MORRISON DAM AND OUTLET WORKS

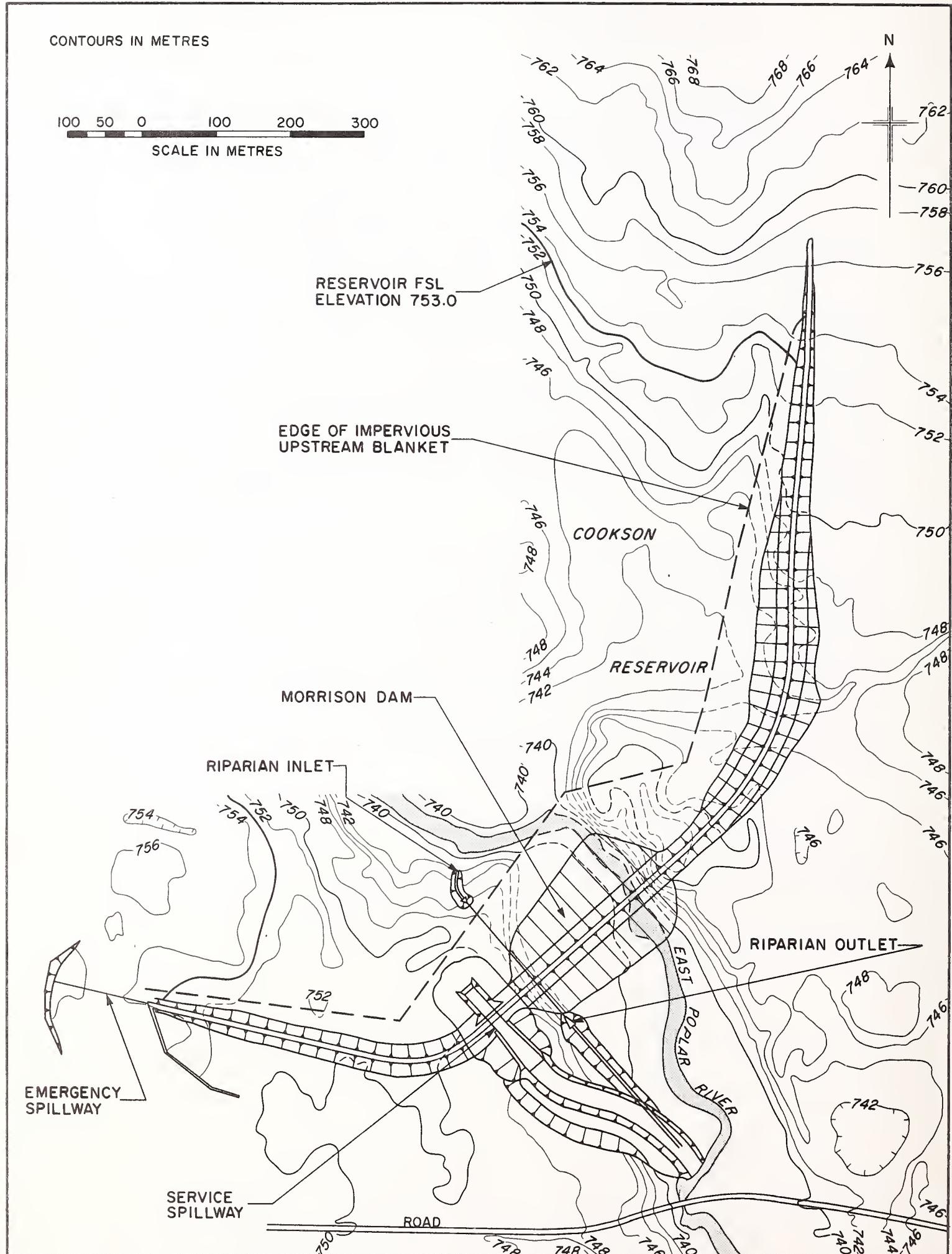
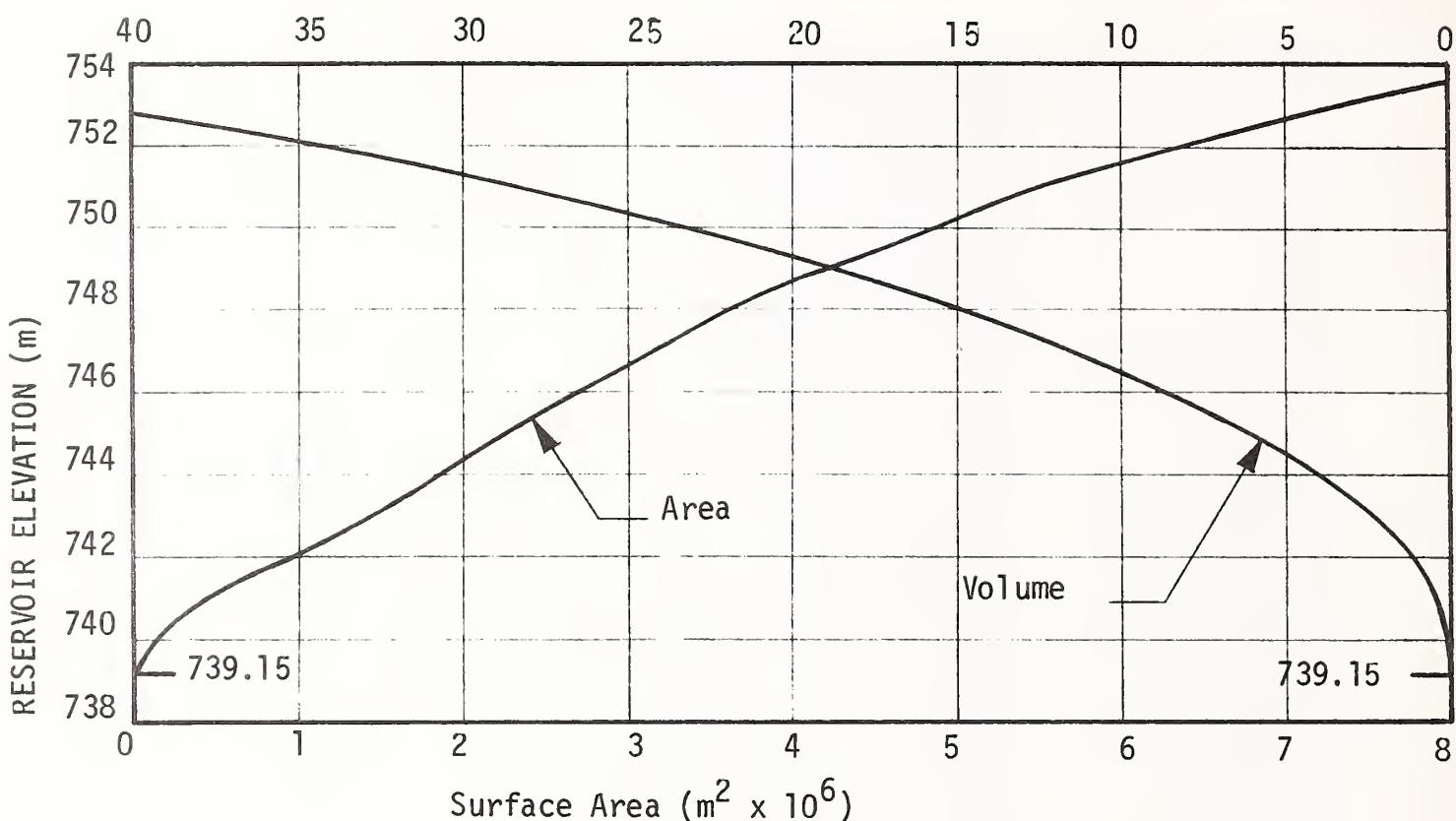


PHOTO 6-1 MORRISON DAM SHOWING SERVICE SPILLWAY AND RIPARIAN OUTLET



FIGURE 6-2 AREA AND CAPACITY DATA FOR COOKSON RESERVOIR

Volume ( $\text{m}^3 \times 10^6$ )

<u>ELEVATION (m)</u>	<u>AREA (<math>\text{m}^2</math>)</u>	<u>VOLUME (<math>\text{m}^3</math>)</u>
739.15 (2425.00)	25,090	0
740. 0 (2427.79)	155,660	78,510
741. 0 (2431.07)	440,880	331,680
742. 0 (2434.35)	1,002,440	1,056,990
743. 0 (2437.63)	1,474,810	2,302,700
744. 0 (2440.91)	1,896,670	3,995,040
745. 0 (2444.20)	2,242,850	6,071,780 (MIN. POOL)
746. 0 (2447.48)	2,708,410	8,535,330
747. 0 (2450.76)	3,188,230	11,489,810
748. 0 (2454.04)	3,582,400	14,871,080
749. 0 (2457.32)	4,183,140	18,731,240
750. 0 (2460.60)	4,857,210	23,253,940
751. 0 (2463.88)	5,485,140	28,419,300
752. 0 (2467.16)	6,358,480	34,300,770
753. 0 (2470.44)	7,368,620	41,166,140 (FSL)
754. 0 (2473.72)	8,417,270	49,054,620

1 acre = 4047  $\text{m}^2$ 1  $\text{m}^2$  = 0.0002471 acres

Although it will substantially reduce the magnitude of spring flows crossing the International Boundary, the reservoir has not been designed and will not be operated for the specific purpose of reducing flood flows downstream. Similarly, although the Poplar River-Nipawin Board of Inquiry (Saskatchewan DOE, 1978) recommended that the land surrounding the reservoir be designated a Reservoir Development Area, it is not expected that any reservoir operation to enhance recreational or other potential uses will be done at the expense of reduced power generation capability.

Since availability of water is a major concern, the basic reservoir operating policy to ensure maximum power generating capability will be to maintain the reservoir as close as possible to the FSL of 753.0 m at all times. The actual operating level will depend on the inflows that have occurred, the number of months since the annual spring freshet inflow, the number of years since the last substantive spring runoff, evaporation losses from the reservoir surface, seepage under and around Morrison Dam and scheduled releases through the outlet works at the dam. There will be no planned surcharge above the FSL for the purpose of containing additional spring runoff water. Any flows that would cause levels to rise above FSL will be passed downstream and regulated by means of the gated service spillway.

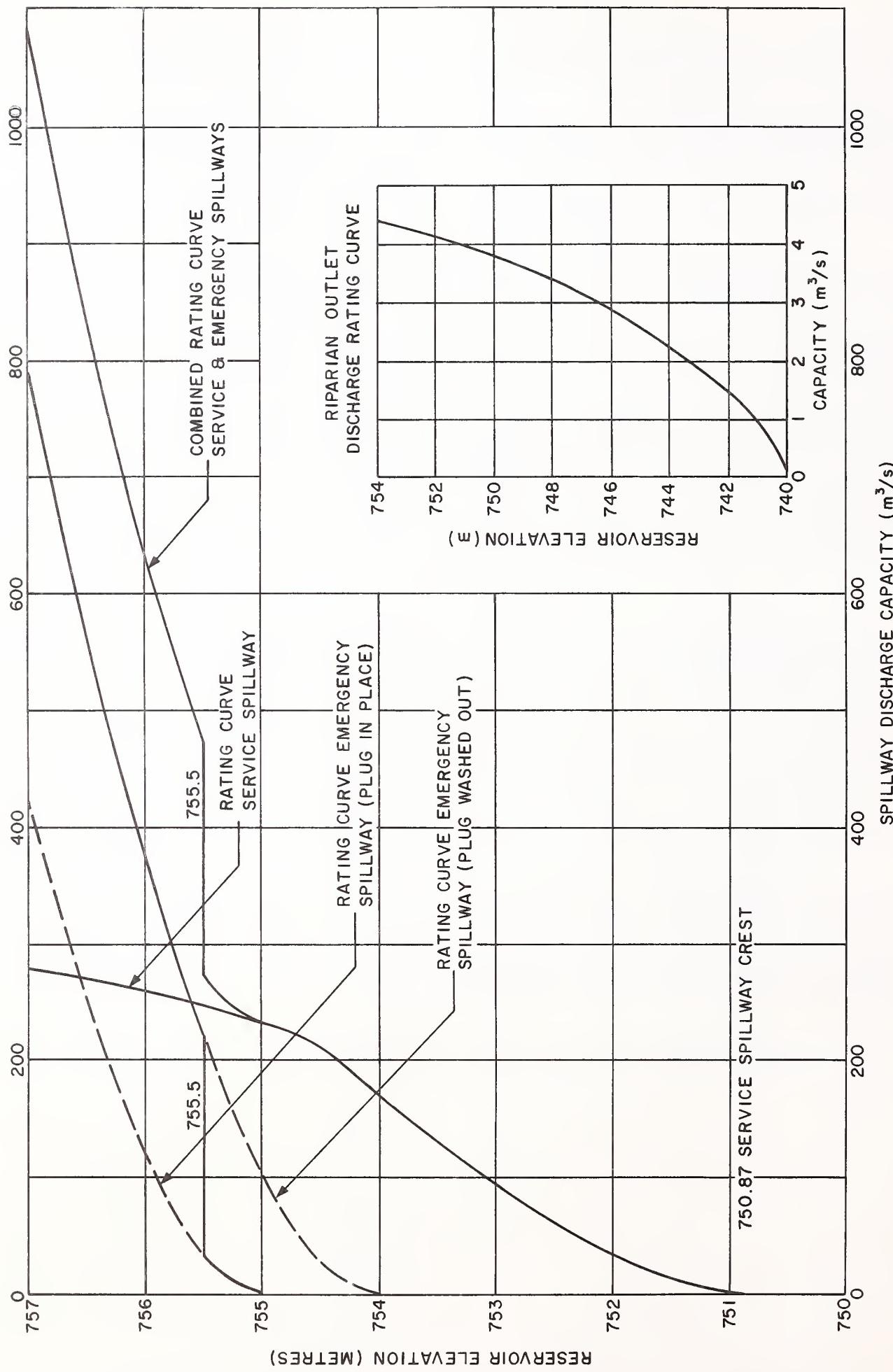
(b) Spillway Release Capacity

Assessment of the capability of releasing flood flows from the reservoir was carried out for the Poplar River Dam Safety Group by the Prairie Farm Rehabilitation Administration (PFRA, 1977) using the discharge rating curves for the service and emergency spillways shown in Figure 6-3. The Plant, Mine and Reservoir Operations Committee was not charged with evaluating the adequacy of the spillway capacity but was requested to provide technical information on reservoir operating constraints. Therefore, only available capacity data is provided here and evaluation of the adequacy of release capacity should be obtained from the Dam Safety Group.

Comparison of the PFRA rating curves in Figure 6-3 with rating curves obtained from the SPC, shows that the PFRA capacity estimates for the service spillway agree with SPC rating curves but for the emergency spillway are somewhat lower than estimates by the SPC. Also the 722 m<sup>3</sup>/s probable maximum flood determined by the PFRA in their analysis is 184 m<sup>3</sup>/s higher than the SPC's estimate of 538 m<sup>3</sup>/s.

Some maximum spillway release rates extracted from the PFRA rating curves in Figure 6-3 are listed below:

FIGURE 6-3 DISCHARGE RATING CURVES – MORRISON DAM SPILLWAY AND RIPARIAN OUTLET



Reservoir Level (m)	Release Capacity		
	Service Spillway (m <sup>3</sup> /s)	Emergency Spillway (m <sup>3</sup> /s)	Total (m <sup>3</sup> /s)
750.87 (crest service spillway)	0	0	0
753.00 (FSL)	93	0	93
755.00 (crest earth- fill plug)	232	0	232
755.30 (1:500 year flood)	240	14	254
755.50 (earthfill plug washout)	246	224	470
756.22 (probable maximum flood)	262	460	722

With the 1:500 year flood, the rating curves show that the reservoir would surcharge 2.3 m to elevation 755.3 m which is 1.7 m below the dam crest elevation of 757.0 m. The earthfill plug would remain in place unless the reservoir level exceeded approximately 755.3 m when it would wash out. With the probable maximum flood and earthfill plug eroded the reservoir would surcharge 3.2 m to elevation 756.2 m or 0.8 m below the crest of the dam.

#### (c) Low-Flow Release Capacity

Low-flow releases will depend upon the apportionment border flows eventually agreed to by the Canadian and United States Governments. The apportionment flows recommended in the IJC report (IJC, 1978) to governments are to be examined by the IPRWQB for their affect on border water quality and it is conceivable that alterations to the apportionment recommendations could be suggested. The recommendations for the East Poplar River as they now stand are as follows:

March-April-May Inflow Volume m <sup>3</sup> x10 <sup>6</sup>	Continuous Release (m <sup>3</sup> /s)	Release on Demand (m <sup>3</sup> )
- 4.69	0.028	370,000
4.69 - 9.25	0.057	617,000
9.25 - 14.80	0.085	617,000
14.80	0.085	1,230,000

Low-flow releases will normally be made using the riparian outlet at Morrison Dam. Whenever the reservoir level is below the service spillway crest elevation of 750.87, the riparian outlet is the only

facility capable of releasing water downstream from the reservoir although water could be pumped over the dam if the riparian outlet was inoperable.

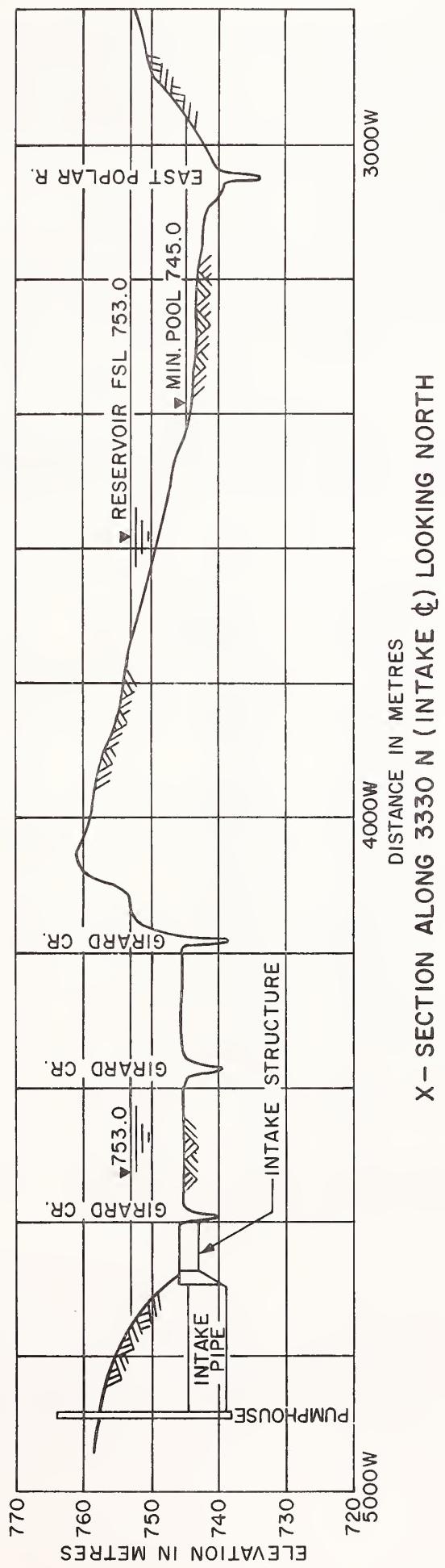
The riparian outlet is a concrete lined corrugated metal pipe beneath Morrison Dam. Flow through the pipe is controlled by means of a gatewell and gate near the centre of the dam. The pipe has a diameter of approximately 1.2 m, and invert elevations of 741.96, 741.76 and 740.14 m at the inlet, gatewell and outlet respectively. The discharge capacity of the riparian outlet with the slide gate fully open is shown on Figure 6-3. The curve shows a capacity of 2.5 m<sup>3</sup>/s at the reservoir minimum pool elevation of 745.0 which would allow the IJC's recommended release on demand volumes of 370,000, 617,000 and 1,230,000 m<sup>3</sup> to be released in 1.7, 2.8 and 5.5 days respectively. When the 617,000 and 1,230,000 m<sup>3</sup> releases are required, which is during average or high runoff years, it is likely that the reservoir level would be well above minimum pool and the riparian outlet capacity would be greater than 2.5 m<sup>3</sup>/s. At reservoir FSL, approximately 4.25 m<sup>3</sup>/s can be released through the outlet. If the service spillway were used, the maximum release of 1,230,000 m<sup>3</sup> can be made in less than 1/2 hour when the reservoir is at FSL.

#### 6.1.3 Circulation and Mixing

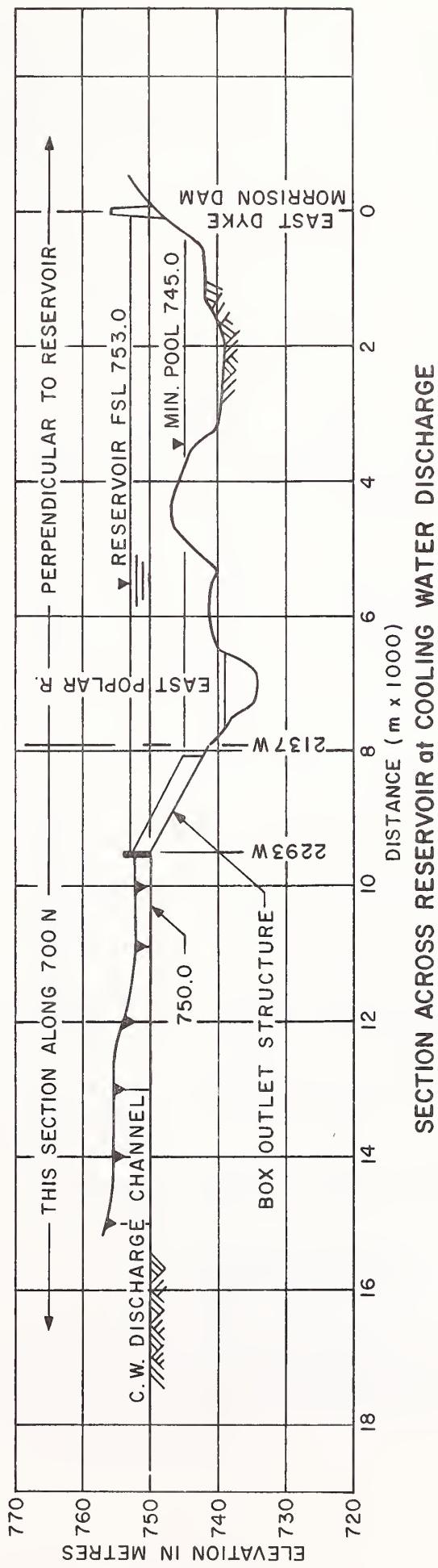
The cooling water intake for the Poplar River generating station is located on the Girard Creek portion of the reservoir approximately 4 km upstream from the dam. Cooling water is returned to the reservoir via an outlet structure located 500 m upstream of the dam. Figure 6-4 shows sections across the reservoir along 3330 N, the centreline location of the cooling water intake, and along a line projected from the outlet structure perpendicular to the reservoir. Computations show that the velocity of flow created in the narrow portion of the reservoir by recirculating the cooling water will be as follows:

Reservoir		1 Unit		2 Units	
Level (m)	X-Sect. Area (m <sup>2</sup> )	C.W. Flow (m <sup>3</sup> /s)	Velocity (m/s)	C.W. Flow (m <sup>3</sup> /s)	Velocity (m/s)
753.0	4660	6.70	0.0014	13.40	0.0020
752.0	4160	6.50	0.0016	13.00	0.0031
751.0	3710	6.25	0.0017	12.50	0.0034
750.0	3260	6.00	0.0018	12.00	0.0037
749.0	2850	5.70	0.0020	11.40	0.0040
748.0	2440	5.40	0.0022	16.20	0.0066
747.0	2050	5.05	0.0025	15.15	0.0074
746.0	1670	4.60	0.0028	13.80	0.0083
745.0	1310	8.00	0.0061	12.00	0.0092

FIGURE 6-4 SECTIONS ACROSS COOKSON RESERVOIR



X - SECTION ALONG 3330 N (INTAKE ⚭) LOOKING NORTH



SECTION ACROSS RESERVOIR at COOLING WATER DISCHARGE

Even at the minimum reservoir level of 745.0 the induced current of 0.0092 m/s is much smaller than typical values of wind driven lake currents in the 0.1 to 0.2 m/s range. Hence recirculating cooling water flow would have a negligible effect on the reservoir current patterns. The highway-railroad embankment crossing the upper part of Cookson Reservoir may inhibit total mixing of reservoir water somewhat. However, the oval-shaped culvert beneath the embankment has a height of about 3.5 m and a width of about 6.2 m and should provide for good water movement between the upper and lower portion of the reservoir and ensure fairly uniform distribution of dissolved elements, especially when these are being examined on a monthly basis. The wind induced currents, however, are likely to be different in the two segments and the upper segment is unlikely to make a contribution to dissipation of heat from the recirculating cooling water.

Cookson Reservoir is not expected to be stratified, even during warm summer months. The temperature-depth data listed below was obtained by Environment Canada for Cookson Reservoir in the months of June, July and August, 1978.

<u>Depth (m)</u>	Water Temperature (°C)		
	<u>June</u>	<u>July</u>	<u>August</u>
0.0	17.0	21.5	19.0
1.0	17.0	21.5	19.0
2.0	16.5	21.5	19.0
3.0	16.0	21.0	19.0
4.0	16.0	21.0	19.0
5.0	16.0	21.0	19.0
6.0	15.5	20.5	19.0
7.0	15.5	20.5	19.0
8.0	15.5	20.5	19.0
9.0	15.5	20.5	19.0
10.0	15.5	20.0	19.0
11.0	15.5	20.0	19.0
12.0	15.5	20.0	19.0
13.0			19.0

These measurements show that wind driven currents are sufficient to prevent reservoir stratification, a condition which has also been observed at the deeper Boundary Dam Reservoir in south eastern Saskatchewan.

#### 6.1.4 Thermal Regime

The Plant, Mine and Reservoir Operations Committee has not made its own analysis of the expected thermal regime in Cookson Reservoir. However, computations of the temperature distribution in the reservoir with 1 and 2 units operating with and without the influence of wind have been

made for the SPC (Spraggs, 1977) using both the general formulation developed by Edinger (Edinger et al, 1974) and a vertically integrated two-dimensional model. Calculations were done for the reservoir at FSL with subsequent estimates made for lower reservoir levels. Assumed cooling water flows were  $7.56 \text{ m}^3/\text{s}$  for 1 unit and  $15.12 \text{ m}^3/\text{s}$  for 2 units, and a temperature rise from inlet to outlet of  $10^\circ\text{C}$  was used.

Spraggs results computed for the reservoir zones shown in Figure 6-5 are provided in Table 6-1 for both the 1 and 2 unit cases. Spraggs assumption that zones 1 to 5 would have the majority of the heat dissipation load with 1 unit operating and zones 1 to 6 with 2 units appears reasonable based on the results of his analysis.

Table 6-1 indicates that average reservoir water temperatures of  $27.66$  and  $30.6^\circ\text{C}$  with 1 and 2 units respectively would occur in Zone 1 near the outfall in July. These temperatures are, of course, dependent on the assumptions made for the cooling water flow rate and condenser temperature rise. The discussion on condensers in Section 5.2.2 suggests that the natural reservoir temperatures could exceed  $25^\circ\text{C}$  in summer and the condenser flow rates would be less than the  $7.56 \text{ m}^3/\text{s}$  per unit assumed by Spraggs whenever the cooling system is operated with only 1 pump per unit or the reservoir level drops below 748 m. This means that with an estimated  $2^\circ\text{C}$  temperature drop in the cooling water discharge channel, cooling water could be released into the reservoir at temperatures ranging from  $34$  to  $40^\circ\text{C}$  and higher water temperatures can be expected in the various reservoir zones than those listed in Table 6-1.

Of most interest to aquatic biology in the East Poplar River downstream of Cookson Reservoir may be the difference between the temperature of water released from the reservoir and natural water temperatures in the river. When spillway releases are required at rates from 0 to  $17 \text{ m}^3/\text{s}$ , the majority of the released water could be heated condenser discharge water buoying up from the cooling water outlet immediately in front of Morrison Dam. It is, hence, conceivable that the water being released down the river could be  $10$  to  $15^\circ\text{C}$  warmer than normal reservoir water temperature for that time of year. When spillway releases greater than  $17 \text{ m}^3/\text{s}$  occur, the release will be greater than the maximum possible cooling water discharge for 2 units and a reduction in the discharge temperature would occur through dilution of the cooling water with the additional water passing through the spillway.

#### 6.1.5 Softening of Reservoir Water

One of the methods that has been suggested as a possibility for controlling condenser tube scaling is cold lime softening of the reservoir water to lower the calcium alkalinity. Since the present quality of water in Cookson reservoir is better than expected, scaling is not expected to be a problem during the first years of plant operation and no decision on the scale control option will be needed for several years. However, since lime softening remains a future possibility and would alter reservoir water quality, there is a need to examine its probable effects.

FIGURE 6-5 RESERVOIR ZONES FOR COMPUTING THERMAL REGIME

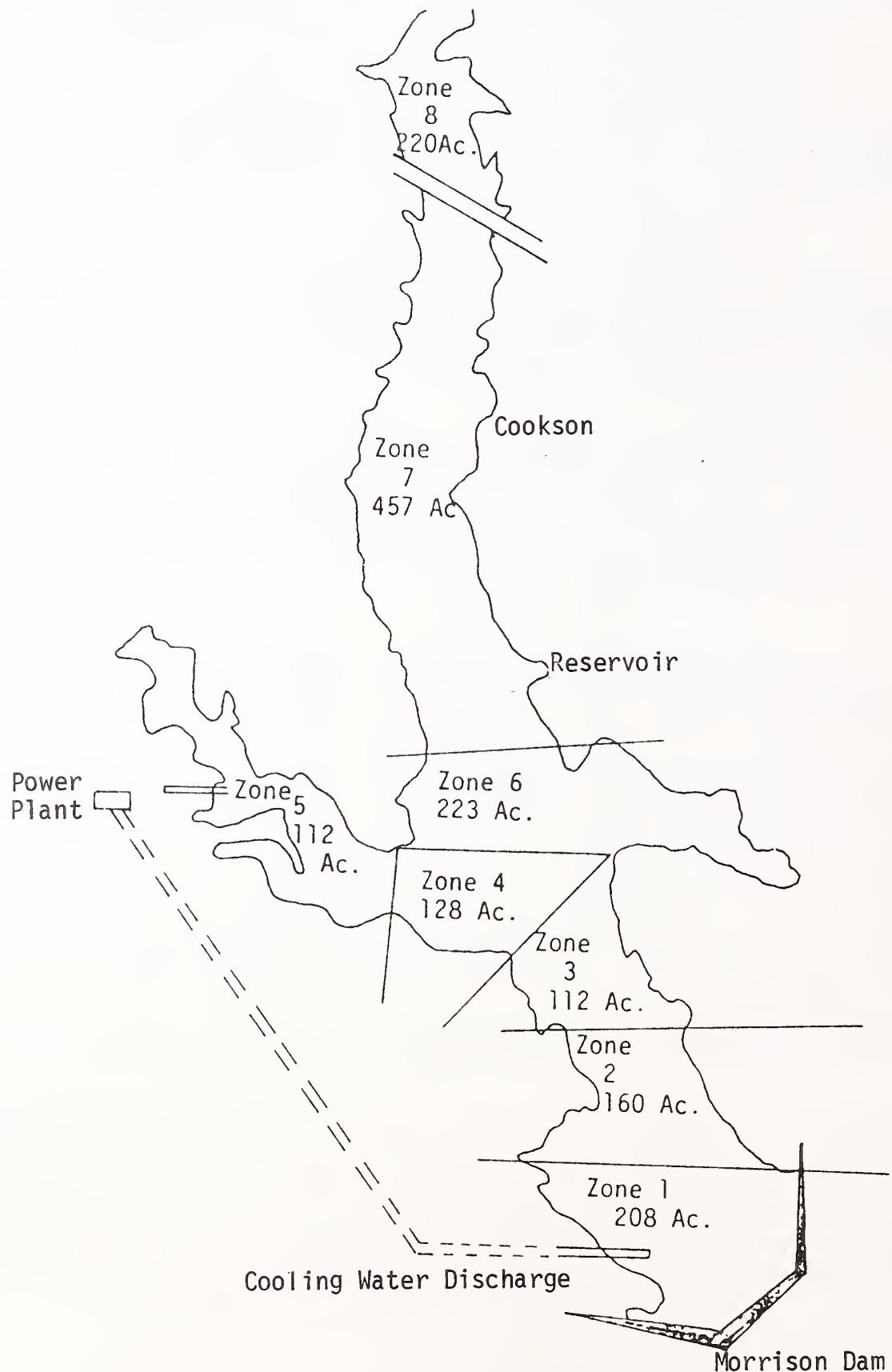


TABLE 6-1 TEMPERATURE DISTRIBUTION IN COOKSON RESERVOIR

DATE	(°C)	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5
<u>1 UNIT</u>						
JAN.	0	3.04	1.64	.88	.34	.06
FEB.	0	3.04	1.64	.88	.34	.06
MAR.	0	5.06	2.74	1.47	.57	.11
APR.	5.6	10.66	8.34	7.07	6.17	5.71
MAY	12.5	17.56	15.24	13.97	13.07	12.61
JUNE	17.0	22.06	19.74	18.47	17.57	17.11
JULY	22.6	27.66	25.34	24.07	23.17	22.71
AUG.	17.9	22.96	20.64	19.37	18.47	18.01
SEPT.	13.0	18.06	15.74	14.47	13.57	13.11
OCT.	6.37	11.43	9.11	7.84	6.94	6.48
NOV.	1.0	6.06	3.74	2.47	1.57	.11
DEC.	0	3.04	1.64	.88	.34	.06
<u>2 UNITS</u>						
JAN.	0	6.6	3.6	2.2	1.5	1.0
FEB.	0	6.6	3.6	2.2	1.5	1.0
MAR.	0	8.0	6.1	4.7	2.5	1.1
APR.	5.6	13.6	11.7	10.3	8.1	6.7
MAY	12.5	20.5	18.6	17.2	15.0	13.6
JUNE	17.0	25.0	23.1	21.7	19.5	18.1
JULY	22.6	30.6	28.7	27.3	25.1	23.7
AUG.	17.9	25.9	24.0	22.6	20.4	19.0
SEPT.	13.0	21.0	19.01	17.7	15.5	14.1
OCT.	6.37	14.3	12.5	11.1	8.9	7.4
NOV.	1.0	9.0	7.1	5.7	3.5	2.1
DEC.	0	6.6	3.6	2.2	1.5	1.0

The lime softening process is described in Annex H. The process requires the addition of significant quantities of sulphuric acid and hydrated lime to effect the removal of calcium alkalinity. The quantities of acid and lime required during any given time period depend on the reservoir water chemistry during that period. In the process of precipitation, Ca, Mg, total alkalinity and P would be removed from solution. TDS reduction would depend on the balance between a sulphate increase due to the sulphuric acid addition and the reduction in dissolved materials by precipitation.

The lime softening process has been incorporated into the reservoir operational model and water quality changes from the results of model runs are described in Section 8 on mitigation.

#### 6.1.6 Recreational Uses

A recreation development plan for Cookson Reservoir (Saskatchewan Department of Tourism and Renewable Resources, 1977) was prepared recognizing that "Recreation in any form is but a secondary use of the reservoir after the production of electric power" and should not restrict or interfere with power production. Some of the plan's recommendations are:

1. Recreation development be restricted to an area lying east of the East Poplar River reservoir;
2. Recreation development be limited to opportunities and facilities for camping, swimming, picnicking, boating, hiking, scenic viewing and baseball;
3. Cottage subdivisions not be permitted adjacent to the East Poplar River reservoir;
4. Appropriate rehabilitative measures be undertaken to enhance the visual quality of the existing borrow pit areas south of Highway No. 36 on the east side of the reservoir;
5. Land lying within the East Poplar River reservoir and plant boundary be declared a wildlife refuse;
6. Recreation development on the reservoir be administered in such a fashion as to provide recreation facilities responsive to local requirements, and
7. People be charged a fee for the use of the reservoir's recreation facilities.

The plan recommended the development of two scenic viewpoints, a 30 site picnic area, a baseball field, a 3.5 mile hiking launch, a playground and a 45 site campground near the southeast corner of the

reservoir. Also recommended was that all sanitary facilities associated with the recreation development be equipped with pump-out vaults and that pump-out liquid wastes be deposited in a nearby municipal treatment system (Coronach).

The recreation development plan was followed by a recommendation in the Poplar River - Nipawin Board of Inquiry report (Saskatchewan DOE, 1978) that the "land surrounding the reservoir be designated a Reservoir Development Area." This recommendation assumes that there will be demands for various types of use of the shoreline and peripheral area of the reservoir. Uses such as those outlined in the recreational development plan can therefore be expected.

## 6.2 Inputs and Losses to Surface Water Systems

### 6.2.1 General

This section deals with the water inputs to and losses from Cookson Reservoir and chemical additions that have not been described as a power plant or mine effluent in Sections 3 and 5. The inputs can be summarized as follows:

- (i) Natural inflow from the East Poplar River and Girard Creek;
- (ii) Leachates from the soil inundated by the reservoir;
- (iii) Chemicals added to the reservoir in the lime softening process;
- (iv) Overflows from Fife Lake.

Water losses from the reservoir include:

- (i) releases through the riparian outlet and spillway at Morrison Dam;
- (ii) Seepage through and beneath Morrison Dam;
- (iii) Leakage around Morrison dam through the Empress Group sands and gravels;
- (iv) Natural evaporation losses;
- (v) Forced evaporation losses.

Each of the above inputs and losses are described in the following sections.

## 6.2.2 Inputs to Cookson Reservoir

### (a) Natural Inflow

The PFRA estimated 1975 present use flows for the East Poplar River at the International Boundary for the period 1931-76 for the IJC water apportionment study (IJC, 1978). These flows, presented in Table 6-2, were determined by deducting consumptive water uses existing in the Canadian portion of the basin in 1975 from the recorded or estimated natural flows at the International Boundary. The flows in Table 6-2 are considered the best available estimate of pre-Poplar River power development flows.

For use in the reservoir operation model, which is described under mitigation in Section 8, a further adjustment has been made to the International Boundary flows listed in Table 6-2 to obtain flows considered more representative of actual inflows to Cookson Reservoir. With the expected post-reservoir construction pattern of groundwater flows, a portion of the naturally occurring East Poplar River base flow at the International Boundary can be expected to enter the reach of the river between Morrison Dam and the boundary rather than discharge into Cookson Reservoir. This will occur partly because some of the baseflow at the boundary entered that portion of the river from the groundwater system under natural conditions and partly because the groundwater table mound built up around the reservoir will tend to reduce direct groundwater flow to the reservoir diverting more of the flow around the reservoir to the river downstream. On the other hand, a portion of the groundwater flow that naturally moved southeastward from the Fife Lake area to enter the west side of the East Poplar River may be cut off by the groundwater table depression cone caused by mine dewatering.

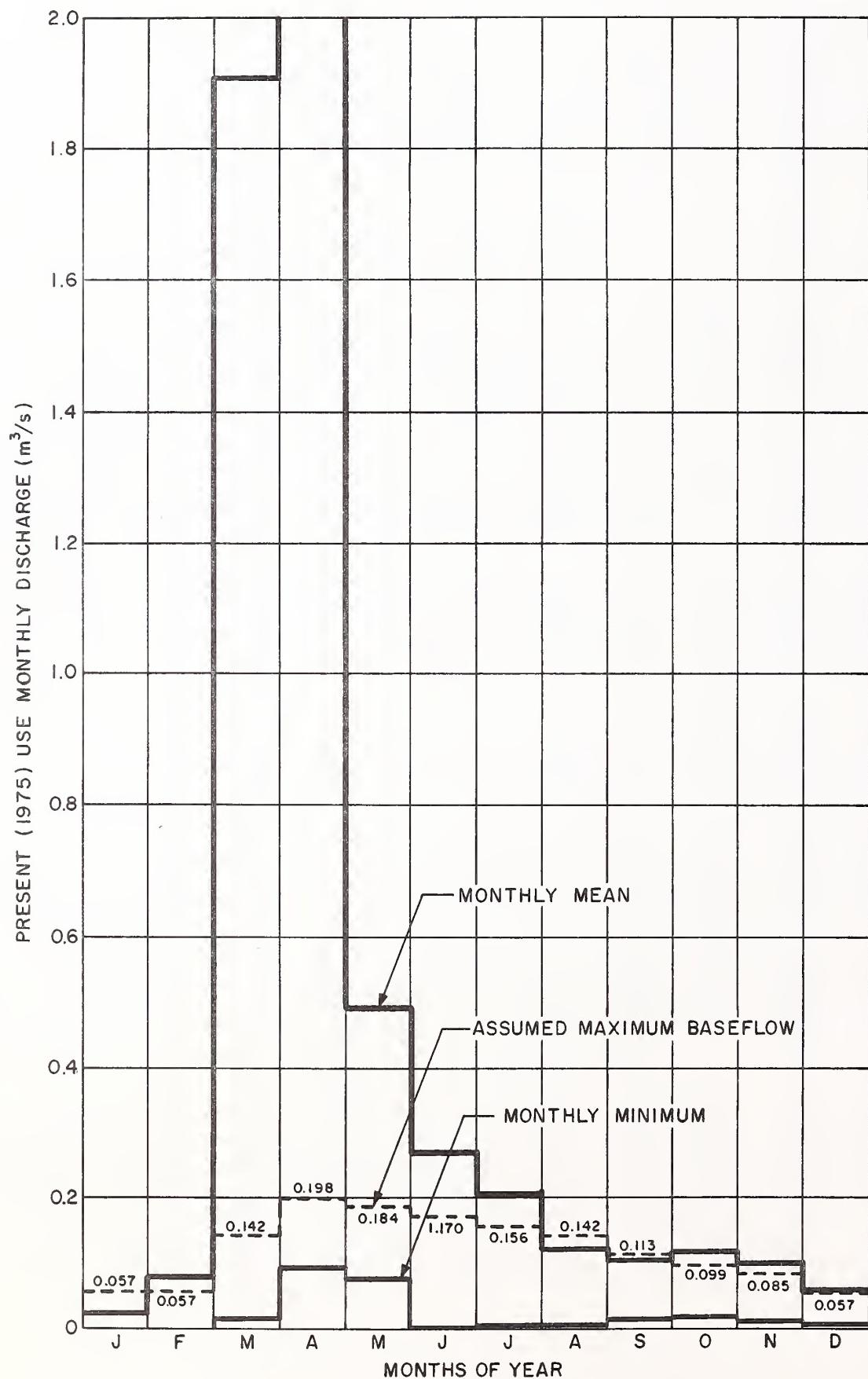
The assumption made for use in reservoir modelling is that half the present use baseflows in the East Poplar River at the International Boundary would enter Cookson Reservoir and half would enter the river between Morrison Dam and the boundary. This assumption implies that inflow to Cookson Reservoir would be somewhat less than that used in the apportionment study making the computations of reservoir levels and water quality more conservative. It also means that a certain amount of groundwater dilution capability is assumed to exist between Morrison Dam and the boundary, that planned releases from the reservoir needed to meet apportionment flows at the boundary are less and that during some flow conditions more water than needed to meet the apportionment low-flow requirements will be discharged across the boundary.

To make the above adjustment to reservoir inflows, an estimate of baseflow is required for each month in the 46 year study period. An assessment of the mean annual hydrograph of East Poplar River flows shown in Figure 6-6 was carried out to establish the following upper limits to the 1975 present use baseflow rates at the International Boundary:

Table 6.2 PRESENT USE (1974) FLOW IN EAST POPLAR RIVER AT INTERNATIONAL BOUNDARY  
(Flow in m<sup>3</sup>/s)

	Jan	Feb	Mar	Apr	May	June	July	Aug	Sept	Oct	Nov	Dec	Mean	%	Volume m <sup>3</sup> x10 <sup>3</sup>
1931	0.025	0.083	0.187	0.093	0.091	0.020	0.020	0.014	0.096	0.102	0.100	0.058	0.074	15	2332.
1932	0.028	0.071	0.909	0.473	0.122	0.059	0.042	1.252	0.096	0.133	0.100	0.045	0.280	55	8845.
1933	0.028	0.038	0.634	0.360	0.394	0.428	0.187	0.079	0.048	0.040	0.073	0.044	0.197	39	6213.
1934	0.039	0.117	0.283	0.306	0.076	0.011	0.003	0.005	0.034	0.045	0.060	0.045	0.085	17	2680.
1935	0.017	0.008	1.260	0.193	0.130	0.300	0.164	0.037	0.023	0.020	0.025	0.011	0.184	36	5814.
1936	0.011	0.018	0.235	1.058	0.210	0.048	0.023	0.025	0.031	0.031	0.037	0.025	0.145	28	4592.
1937	0.025	0.025	0.062	0.150	0.207	0.062	0.122	0.023	0.037	0.088	0.088	0.060	0.080	16	2508.
1938	0.032	0.033	3.795	0.269	0.235	0.125	0.144	0.028	0.057	0.088	0.085	0.044	0.417	82	13163.
1939	0.023	0.018	6.172	0.187	0.147	0.510	0.034	0.008	0.031	0.031	0.066	0.054	0.616	121	19427.
1940	0.008	0.029	0.127	1.663	0.229	0.144	0.108	0.286	0.085	0.116	0.082	0.038	0.241	47	7634.
1941	0.018	0.043	2.916	0.303	0.176	0.269	0.150	0.068	0.085	0.105	0.114	0.061	0.363	71	11460.
1942	0.039	0.069	2.420	1.130	0.144	0.133	0.161	0.093	0.320	0.198	0.142	0.102	0.415	81	13100.
1943	0.022	0.050	8.675	1.645	0.159	0.278	0.139	0.034	0.059	0.110	0.114	0.066	0.958	188	30209.
1944	0.029	0.024	0.085	0.286	0.136	0.201	0.085	0.122	0.059	0.074	0.072	0.021	0.099	19	3140.
1945	0.018	0.032	1.111	0.252	0.178	0.125	0.057	0.074	0.119	0.071	0.081	0.060	0.183	36	5771.
1946	0.022	0.018	2.528	0.244	0.122	0.108	0.144	0.017	0.023	0.085	0.102	0.039	0.292	57	9196.
1947	0.008	0.002	2.509	4.756	0.232	0.408	0.105	0.116	0.105	0.125	0.086	0.050	0.707	139	22305.
1948	0.018	0.016	0.314	8.739	0.493	0.127	0.105	0.082	0.071	0.071	0.088	0.029	0.835	164	26407.
1949	0.001	0.002	1.659	1.113	0.159	0.096	0.042	0.042	0.085	0.093	0.091	0.033	0.286	56	9032.
1950	0.0	0.000	0.051	6.742	0.306	0.0	0.116	0.122	0.113	0.122	0.092	0.048	0.636	125	20057.
1951	0.004	0.008	0.076	1.892	0.767	0.164	0.125	0.113	0.184	0.164	0.097	0.050	0.303	59	9556.
1952	0.001	0.000	0.068	16.628	0.634	0.144	0.153	0.105	0.127	0.159	0.104	0.065	1.494	293	47246.
1953	0.033	0.040	0.487	0.430	0.365	0.983	0.674	0.113	0.136	0.167	0.132	0.086	0.305	60	9609.
1954	0.034	0.197	0.892	8.453	2.362	0.827	0.144	0.227	0.337	0.204	0.170	0.109	1.157	227	36479.
1955	0.037	0.024	2.710	9.313	4.437	0.210	0.153	0.085	0.110	0.144	0.089	0.033	1.446	284	45612.
1956	0.002	0.002	0.971	0.801	0.269	0.161	0.082	0.017	0.062	0.125	0.113	0.060	0.223	44	7041.
1957	0.023	0.027	0.348	0.334	0.190	0.147	0.093	0.079	0.136	0.221	0.192	0.129	0.161	31	5065.
1958	0.086	0.137	2.367	0.804	0.156	0.133	0.122	0.068	0.119	0.133	0.103	0.048	0.359	70	11320.
1959	0.001	0.002	0.311	0.241	0.161	0.108	0.091	0.076	0.099	0.161	0.122	0.086	0.122	24	3859.
1960	0.022	0.029	6.756	0.292	0.164	0.150	0.190	0.105	0.113	0.125	0.108	0.067	0.686	134	21686.
1961	0.046	0.067	0.912	0.193	0.210	0.119	0.065	0.034	0.130	0.102	0.084	0.044	0.168	33	5310.
1962	0.002	0.006	3.081	0.821	0.193	0.419	0.328	0.093	0.110	0.150	0.117	0.087	0.455	89	14356.
1963	0.025	0.558	2.263	0.459	0.246	0.377	0.238	0.119	0.133	0.113	0.097	0.060	0.391	77	12319.
1964	0.023	0.044	0.144	1.297	0.184	0.156	0.096	0.048	0.113	0.119	0.087	0.033	0.194	38	6131.
1965	0.002	0.001	0.016	2.605	0.547	0.303	0.232	0.076	0.164	0.184	0.129	0.081	0.360	71	11349.
1966	0.000	0.001	1.642	0.198	0.232	0.136	0.102	0.088	0.116	0.294	0.145	0.061	0.255	50	8026.
1967	0.022	0.018	2.721	4.902	0.402	0.133	0.076	0.031	0.139	0.147	0.107	0.065	0.730	143	23008.
1968	0.004	0.057	3.160	0.263	0.136	0.130	0.150	0.142	0.176	0.127	0.114	0.079	0.382	75	12088.
1969	0.004	0.003	0.025	8.455	0.289	0.110	0.147	0.108	0.099	0.136	0.116	0.081	0.789	155	24883.
1970	0.033	0.046	1.119	3.361	1.413	0.184	0.170	0.062	0.122	0.116	0.094	0.057	0.565	111	17811.
1971	0.017	0.035	1.019	3.729	0.184	0.139	0.082	0.054	0.125	0.133	0.089	0.058	0.470	92	14809.
1972	0.022	0.065	5.094	0.360	0.479	0.289	0.150	0.125	0.150	0.153	0.116	0.058	0.595	117	18816.
1973	0.041	0.067	0.221	0.294	0.235	0.150	0.105	0.042	0.096	0.113	0.099	0.042	0.126	25	3962.
1974	0.043	0.074	3.823	6.150	0.283	0.229	0.102	0.085	0.122	0.136	0.097	0.059	0.933	183	29418.
1975	0.093	0.082	0.261	12.191	1.794	1.180	1.280	0.250	0.121	0.016	0.019	0.017	1.432	281	45157.
1976	0.124	1.219	11.326	6.691	2.318	1.894	2.362	0.740	0.0	0.022	0.164	0.170	2.259	443	71433.
Min	0.0	0.000	0.016	0.093	0.076	0.0	0.003	0.005	0.0	0.016	0.019	0.011	0.074		2332.
Max	0.124	1.219	11.326	16.628	4.437	1.894	2.362	1.252	0.337	0.294	0.192	0.170	2.259		71433.
Mean	0.025	0.077	1.908	2.633	0.491	0.270	0.206	0.122	0.107	0.118	0.100	0.059	0.510	100	16092.

FIGURE 6-6 BASEFLOW DETERMINATION FOR EAST POPLAR RIVER  
AT INTERNATIONAL BOUNDARY



<u>Month</u>	<u>Maximum Baseflow (l/s)</u>
January	57
February	57
March	142
April	198
May	184
June	170
July	156
August	142
September	113
October	99
November	85
December	57

The present use baseflow rates are computed in the reservoir operational model as the lesser of the monthly flows listed in Table 6-2 and the maximum rate listed above. Half the computed baseflow assumed to enter the river downstream of Morrison Dam is then deducted from the total flow to obtain the reservoir inflow for each month.

Inflows to Cookson Reservoir are assumed to have dissolved element concentrations that are inversely proportional to the flow rate. The Surface Water Quality Committee has provided concentration versus flow regression equations for most conservative parameters and boron. Although these may have since been refined, the equations used in reservoir operational model analyses for the purpose of determining and assessing mitigation needs are as follows:

Specific Conductance (EC)	=	$2158 Q^{-0.45}$
Total Alkalinity (as CaCO <sub>3</sub> )	=	0.39 EC + 23.96
Calcium (as CaCO <sub>3</sub> )	=	0.0974 EC + 31.908
Magnesium (as CaCO <sub>3</sub> )	=	0.1605 EC + 12.016
Sodium & Potassium ( as Na)	=	0.142 EC - 18.5
Sulphate (as SO <sub>4</sub> )	=	0.179 EC - 14.8
Boron (as B)	=	0.00139 EC - 0.137

Concentration versus flow data from Table B-5 of the environmental assessment report (SPC, 1978) was used for chloride, total phosphorus and total nitrogen. Total dissolved solids (TDS) is calculated in the reservoir operational model from:

$$TDS = Na + K + Cl + SO_4 + 0.6(T.Alk) + Mg/4.12 + Ca/2.5$$

#### (b) Leachates from Soil Inundated by the Reservoir

When the reservoir rises to flood previously uninundated land, soluble constituents in the newly flooded soil will be taken into

solution in the reservoir water. As the reservoir water level rises and recedes, water will migrate through the soil out of and into the reservoir respectively. The degree of dissolution of soluble components from the soil will depend on this movement, the quantity of water accumulated during successive spring floods, the depth, wind induced wave action, anaerobic conditions developed under ice cover, and subsequent turnover in the spring melt.

To assess this leaching situation, soil samples were collected by Sheppard T. Powell Consultants Limited (Saskmont, 1975) during October and November 1974 from 18 areas in the two valleys which are now flooded. The results of leaching tests performed on these samples are contained in Annex D. 15 additional soil samples were collected around the periphery of the partially filled reservoir by the SPC in April, 1978 at the request of the Plant, Mine and Reservoir Operations Committee. Leaching tests for common ions, trace elements and organics were subsequently performed on these samples at the University of Montana (Pagenkopf, May, 1978). The results of these leachate tests are provided in Annex D and summarized in Table 6-3. Readily leachable salts range from 0.72 to 12.63 kg/m<sup>3</sup> of soil and average 3.56 kg/m<sup>3</sup>. To provide a general appreciation of what this means in terms of addition of salts to the reservoir water, if all readily soluble salts were leached from the 2 m of surface soils over half the reservoir area, the added salts would total  $3.78 \times 7,368,620 = 27,853,000$  kg. If added to the full reservoir water volume of 41,166,140 m<sup>3</sup> this would amount to a total increase in dissolved solids of 680 mg/l.

Since not all salts dissolved in the laboratory testing will necessarily dissolve in actual practice, and the leachate water will migrate out of as well as into the reservoir, the actual quantities ending up in the reservoir may be considerably less than indicated above. Further, it is reasonable to assume that most of the total dissolution of elements from the soil will be realized during the process of reservoir filling or within a short time period thereafter. Therefore, it can be considered a one time occurrence and elements leached from the soil will have contributed to present reservoir water quality. Chemical additions from soil leaching need not be considered as a continuing contributor to reservoir water quality during the period of record being analyzed.

#### (c) Chemical Changes with Lime Softening

If the lime softening process was eventually chosen as a method of condenser scale control, water chemistry in Cookson Reservoir could be substantially altered. The actual additions of lime and sulphuric acid needed to maintain a non-scaling water chemistry are very variable depending on the reservoir water chemistry at any point in time. The formulation for computing the amounts needed is contained in the reservoir operational model, which has been run for the assessments in Section 8 to determine whether lime softening would increase the need for mitigation or be in itself a potential mitigation measure.

Table 6-3 Readily Soluble Boron and Total Salts in Reservoir Area Soil Samples.

<u>Sample Number</u>	<u>Soluble Quantity (g/m<sup>3</sup> of Soil)</u>	<u>Total Salts</u>
Boron		
PR1		1520
PR2	1.23	3440
PR3	0.48	800
PR4	1.05	5630
PR5	0.98	830
PR6	0.60	900
PR7	1.31	3180
PR8	2.06	12630
PR9	1.40	5290
PR10	0.96	4110
PR11	0.35	870
PR12	0.57	720
PR13	1.04	2900
PR14	0.81	4350
PR15	1.26	6240
Mean	1.01	3561

(d) Fife Lake Overflows

Fife Lake forms a large natural storage area in the upper part of Girard Creek Basin. Because of the small quantity of annual inflow and the large surface area for concentration due to evaporation, the quality of water in Fife Lake is generally poor. A small control structure at the outlet of the lake prevents releases downstream into Girard Creek. On the occasional year, some overflow does occur. Fife Lake overflowed both in 1975 and 1976 and both overflows were noted to have a substantial effect on downstream water quality. The 1975 overflow occurred before Morrison Dam was completed and passed on down the East Poplar River as it would have naturally. The 1976 overflow was retained in Cookson Reservoir and contributed to a fairly poor water quality in the reservoir during the first year of reservoir filling.

No firm estimate of the volume of Fife Lake overflow in 1975 or 1976 is available. Quality measurements of Fife Lake water from June to August, 1975 yielded the following:

<u>Parameter</u>	<u>Concentration (mg/l)</u>	<u>Range</u>
<u>Mean</u>		
TDS	1620	1435-1746
Na	459	381-518
K	48	27-83
T. Alk	701	617-765
SO <sub>4</sub>	604	517-657
C <sub>1</sub>	22	20-25
P	0.36	0.33-0.38
NH <sub>3</sub> (as N)	0.34	0.15-0.60
NO <sub>3</sub> (as N)	0.11	0.03-0.27
N (total)	3.0	1.5-4.4
SiO <sub>2</sub>	5.1	3.0-6.6

It is conceivable that water of this quality or worse could overflow from Fife Lake again unless better overflow control is implemented. Although this can be considered a pre-development condition, overflows would now be retained by Cookson Reservoir and contribute to reduced reservoir water quality over an extended time period.

6.2.3 Losses from Cookson Reservoir

(a) Releases at Morrison Dam

Flow releases at Morrison Dam will consist of planned releases to meet apportionment flow requirements or flood flow releases through

the service or emergency spillways during high spring inflow periods. The low-flow releases will be limited to those flows that when added to the water from seepage under and around the dam, seepage from the ash lagoons and groundwater discharge will provide the recommended apportionment flow at the International Boundary. The release through the riparian outlet needed to meet this requirement is computed on a monthly bases in the reservoir operational model. The release-on-demand flow in the IJC recommended apportionment could occur during any month or be split between several months. For the reservoir model runs, it has been assumed to occur in July of each year.

(b) Seepage Through and Beneath Morrison Dam

In an analysis done (Keith Consulting Ltd, 1977) for the Dam Safety Review Committee, it was assumed that with the impervious till core in Morrison Dam and the till-covered upland wing sections that seepage through the dam or through the embankment under the wing dikes would be negligible in comparison to seepage under the core section of the dam. A 1 m thick impervious blanket has been constructed upstream of the dam to lengthen the seepage path and lower the hydraulic gradient. A flow net was constructed and idealized uniform flow calculations were carried out for seepage flow under the impervious blanket. The seepage rate was calculated to be 25.5 l/s (0.9 cfs). Since data gathered on seepage rates since the reservoir has been filled is not sufficient to conclude seepage should be substantially different from 25.5 l/s, this rate has been used in the reservoir operational model. This is nearly equal to the minimum recommended apportionment flow of 28 l/s.

From a quality standpoint, no alteration of dissolved elements along the seepage path beneath the dam is assumed, and elemental concentrations used in the model are the same as the computed monthly concentration in the reservoir water.

(c) Seepage Around Morrison Dam

Estimates were provided to the Plant, Mine and Reservoir Operations Committee by the Groundwater Quantity and Quality Committee of leakage from the reservoir around Morrison Dam through the Empress Group sand and gravel layer. These estimates suggest that the total leakage would be about 7.1 l/s (0.25 cfs) from each side of Cookson Reservoir. However, with the groundwater mound built up by the ash lagoons, the leakage from the west side of the reservoir would be prevented and, in essence, be replaced by ash lagoon seepage flow. Hence, leakage from the reservoir around Morrison Dam to the East Poplar River upstream of the International Boundary is assumed to average 7.1 l/s (0.25 cfs), the contribution from the east side only. A worst case assumption would be no attenuation of dissolved elements and for modelling this component of International Boundary flow is assumed to have reservoir water quality.

(d) Natural Evaporation from Cookson Reservoir

A number of different estimates have been made of both natural and forced evaporation from Cookson Reservoir. These include estimates by:

- (i) Dr. L.D. Spraggs (1977)
- (ii) Dr. M.A. Shirazi (1977)
- (iii) PFRA (1978)

The estimates of gross natural evaporation developed by Dr. Spraggs and provided in Table 6-4 were used in the SPC's environmental impact assessment report. As a first check on these estimates, they were compared on both an annual and a monthly distribution basis with independent estimates by the Prairie Farm Rehabilitation Administration (PFRA). The results of the comparison are presented in Figure 6-7. It can be seen from the graphs in Figure 6-7 that there is reasonable agreement between the Spraggs and PFRA estimates. On a mean annual basis, the PFRA estimates are 8.9 percent lower than those of Spraggs. The PFRA estimates do, however, exceed the Spraggs estimates for 9 of the 46 years, with a maximum annual exceedance of 19.7 percent.

To further assess the validity of the Spraggs estimates and evaluate the consequences of assumptions used, an independent analysis of the Spraggs and Shirazi estimates was undertaken (James F. MacLaren Ltd. and J.E. Edinger Associates Inc, 1978) for the Plant, Mine and Reservoir Operations Committee. Copies of the Shirazi evaporation study and a copy of the MacLaren report are contained in Annex I.

The basic conclusions of the report are that any errors induced by assumptions in the estimates by Dr. Spraggs, or discrepancies observed between his and the other estimates are small in comparison to errors that could be introduced in any of the estimates through incorrect assumptions of the wind speed function and the winter flow guestimates used in the water balance determinations. The report suggests that Spraggs results be used as presented until there is some evidence indicating they are consistently in error in one direction or the other. The report does suggest, however, that the method of deducting precipitation used by Spragg may result in non-conservative net evaporation estimates.

Dr. Spraggs computed net monthly natural evaporation by deducting from the gross evaporation the monthly precipitation in the basin minus the percentage of precipitation known to have occurred as runoff in the East Poplar River during the month. It is reasonable to expect the percentage of the precipitation on the valley area inundated by the reservoir that contributed to the pre-construction gauged flows in the East Poplar River

TABLE 6-4

MONTHLY NATURAL GROSS EVAPORATION (mm)  
FROM COOKSON RESERVOIR

YEAR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
1931	0.	0.	0.	7.	161.	174.	219.	170.	136.	115.	0.	0.
1932	0.	0.	0.	7.	118.	123.	145.	136.	111.	70.	0.	0.
1933	0.	0.	0	7.	85.	137.	228.	179.	153.	94.	0.	0.
1934	0.	0.	0.	7.	175.	159.	195.	206.	115.	63.	0.	0.
1935	0.	0.	0.	7.	99.	116.	149.	248.	133.	110.	0.	0.
1936	0.	0.	0.	7.	132.	152.	203.	200.	144.	102.	0.	0.
1937	0.	0.	0.	6.	131.	209.	197.	233.	138.	98.	0.	0.
1938	0.	0.	0.	5.	130.	140.	215.	218.	169.	124.	0.	0.
1939	0.	0.	0.	8.	139.	115.	205.	238.	163.	106.	0.	0.
1940	0.	0.	0.	4.	133.	169.	171.	267.	150.	111.	0.	0.
1941	0.	0.	0.	7.	135.	192.	201.	196.	136.	98.	0.	0.
1942	0.	0.	0.	8.	112.	130.	162.	155.	130.	109.	0.	0.
1943	0.	0.	0.	10.	118.	125.	237.	241.	178.	97.	0.	0.
1944	0.	0.	0.	6.	139.	127.	191.	178.	145.	129.	0.	0.
1945	0.	0.	0.	6.	113.	106.	221.	215.	142.	93.	0.	0.
1946	0.	0.	0.	8.	125.	126.	190.	203.	129.	80.	0.	0.
1947	0.	0.	0.	5.	128.	99.	243.	189.	156.	87.	0.	0.
1948	0.	0.	0.	5.	111.	131.	198.	190.	184.	139.	0.	0.
1949	0.	0.	0.	9.	134.	156.	217.	229.	198.	99.	0.	0.
1950	0.	0.	0.	6.	121.	146.	172.	158.	152.	85.	0.	0.
1951	0.	0.	0.	6.	141.	133.	188.	172.	123.	72.	0.	0.
1952	0.	0.	0.	6.	123.	178.	193.	202.	151.	150.	0.	0.
1953	0.	0.	0.	6.	120.	93.	210.	226.	167.	116.	0.	0.
1954	0.	0.	0.	6.	111.	136.	195.	144.	117.	96.	0.	0.
1955	0.	0.	0.	5.	115.	162.	168.	228.	160.	100.	0.	0.
1956	0.	0.	0.	6.	126.	186.	176.	186.	153.	100.	0.	0.
1957	0.	0.	0.	5.	143.	148.	210.	179.	153.	8.	0.	0.
1958	0.	0.	0.	5.	162.	179.	193.	223.	167.	96.	0.	0.
1959	0.	0.	0.	7.	131.	184.	216.	227.	153.	70.	0.	0.
1960	0.	0.	0.	6.	138.	185.	217.	212.	194.	114.	0.	0.
1961	0.	0.	0.	6.	129.	236.	246.	266.	183.	95.	0.	0.
1962	0.	0.	0.	8.	131.	182.	220.	208.	177.	97.	0.	0.
1963	0.	0.	0.	6.	135.	148.	227.	220.	178.	144.	0.	0.
1964	0.	0.	0.	7.	170.	172.	261.	247.	143.	110.	0.	0.
1965	0.	0.	0.	5.	160.	148.	200.	208.	127.	108.	0.	0.
1966	0.	0.	0.	6.	172.	162.	215.	217.	166.	122.	0.	0.
1967	0.	0.	0.	5.	135.	184.	274.	235.	204.	100.	0.	0.
1968	0.	0.	0.	8.	162.	169.	234.	187.	133.	102.	0.	0.
1969	0.	0.	0.	6.	150.	191.	185.	268.	169.	81.	0.	0.
1970	0.	0.	0.	5.	130.	200.	207.	259.	169.	106.	0.	0.
1971	0.	0.	0.	6.	194.	170.	228.	276.	184.	115.	0.	0.
1972	0.	0.	0.	8.	119.	189.	173.	190.	154.	86.	0.	0.
1973	0.	0.	0.	4.	127.	197.	237.	235.	152.	102.	0.	0.
1974	0.	0.	0.	5.	102.	189.	219.	156.	129.	107.	0.	0.
1975	0.	0.	0.	4.	86.	151.	206.	178.	135.	83.	0.	0.
1976	0.	0.	0.	6.	188.	147.	191.	214.	171.	107.	0.	0.

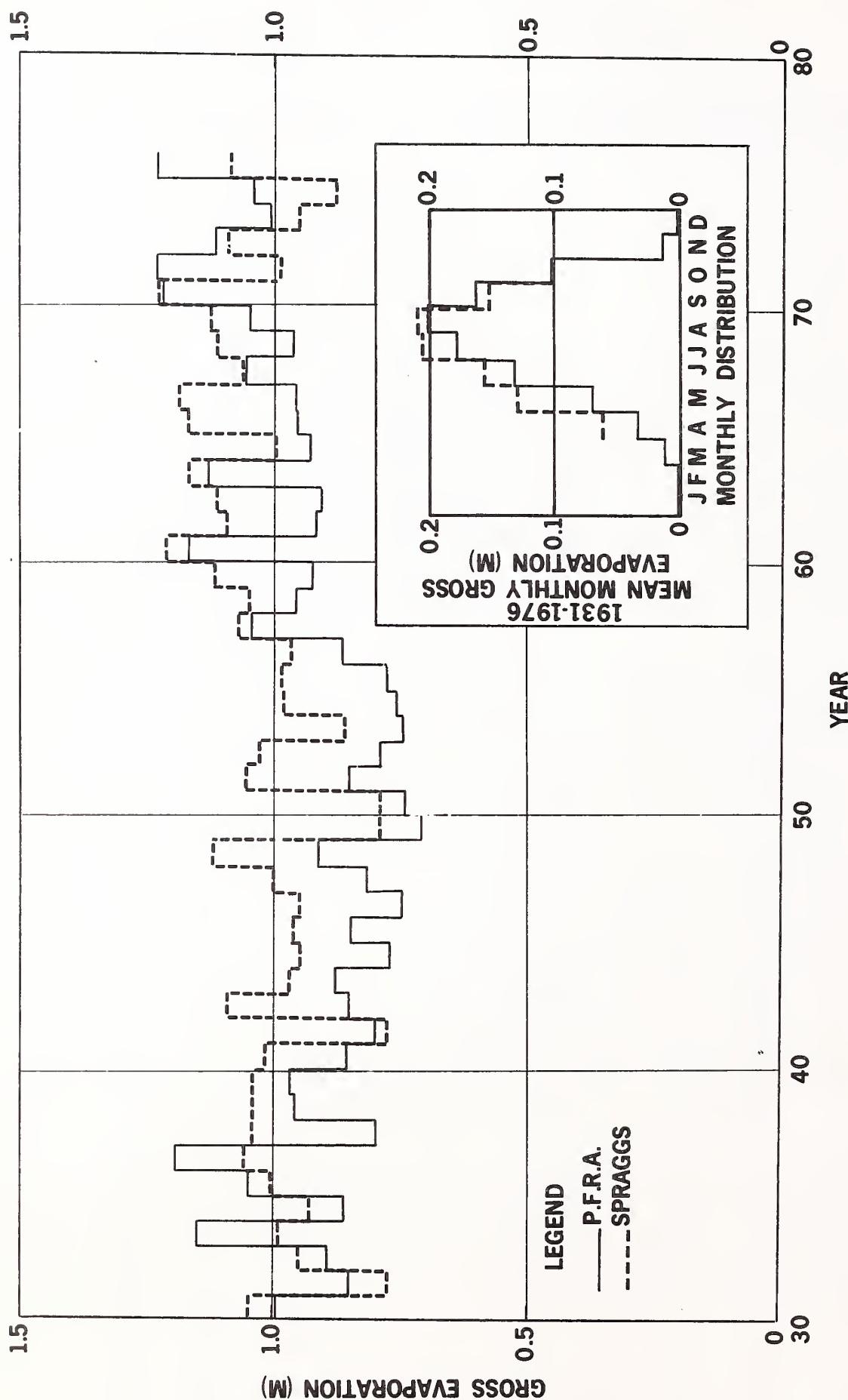


FIGURE 6-7 GROSS EVAPORATION ESTIMATES FOR COOKSON RESERVOIR 1931-1976

would have been greater than the basin wide percentage contribution. Because of the proximity of the valley area to the river channel, rainfall there would have short travel times and traverse steeper slopes relative to much of the rest of the basin. A very crude analysis shows what the percent contributions to the mean annual flow ( $16,092,000 \text{ m}^3$ ,  $0.510 \text{ m}^3/\text{s}$ ) in the East Poplar River would be with different percent contributions from the mean annual precipitation ( $0.37 \text{ m}$ ) in the reservoir area ( $7,368,620 \text{ m}^2$ ).

<u>% Reservoir Area Precipitation Contributing to runoff</u>	<u>Contributing volume from Reservoir Area (m<sup>3</sup>)</u>	<u>% of Mean Annual Flow</u>
100	2,726,390	17
50	1,363,190	8
10	272,640	2
2	54,527	0.3

Even at 100% contribution from the reservoir area, the precipitation in the immediate Poplar River valley would have made up only 17% of the flow in the river. If 100% of the valley precipitation did contribute to runoff, the average contribution from the remainder of the drainage basin would change only slightly to 4.7% instead of the basin wide average of 5.6%.

It is likely that considerably more than 5.6% of the precipitation on the valley area would have contributed on average to runoff and be included in the East Poplar River flow records, especially in light of the fact that snow would tend to blow into the valley from the surrounding flatter land. However, the evapotranspiration from the vegetation and land surface in the valley prior to creation of the reservoir would likely have averaged in the order of  $0.4 \text{ m}$  per year for the region (Morton, 1978), which is roughly equivalent to the mean annual precipitation of  $0.37 \text{ m}$ . The water being lost to evapotranspiration would be replenished by a fairly continuous supply from groundwater discharge in the immediate vicinity of the river. In the environmental assessment report (SPC, 1978), evapotranspiration losses were not deducted in determining net evaporation losses from the reservoir. Since an over estimation of the precipitation and omission of the evapotranspiration deduction would tend to cancel errors in the net evaporation calculation, it is recommended that net evaporation from Cookson Reservoir be computed by deducting the precipitation figures in Table 6-5 from the gross evaporation in Table 6-4 pending further refinement.

#### (e) Forced Evaporation from Cookson Reservoir

Forced evaporation refers to the additional evaporative water loss from the reservoir resulting from the addition of heat from the

TABLE 6-5

MONTHLY PRECIPITATION (mm) IN EAST POPLAR RIVER BASIN

YEAR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
1931	3.	2.	17.	6.	17.	65.	30.	55.	60.	4.	14.	33.
1932	22.	9.	15.	34.	21.	67.	152.	95.	8.	18.	23.	18.
1933	15.	22.	8.	27.	56.	73.	17.	118.	47.	23.	11.	38.
1934	18.	4.	13.	3.	10.	124.	36.	12.	48.	7.	4.	13.
1935	42.	8.	13.	39.	29.	92.	86.	48.	1.	22.	24.	13.
1936	14.	16.	13.	13.	31.	67.	16.	25.	19.	15.	6.	13.
1937	17.	10.	5.	5.	31.	12.	25.	21.	23.	20.	17.	20.
1938	23.	45.	31.	12.	52.	59.	27.	38.	44.	22.	11.	12.
1939	16.	14.	16.	17.	82.	137.	55.	7.	8.	24.	1.	17.
1940	8.	20.	13.	35.	22.	80.	42.	6.	41.	18.	15.	7.
1941	11.	4.	13.	6.	24.	55.	45.	79.	23.	1.	23.	16.
1942	11.	20.	10.	62.	19.	167.	47.	74.	48.	38.	22.	14.
1943	25.	13.	15.	2.	77.	43.	27.	28.	29.	56.	5.	4.
1944	7.	17.	28.	1.	113.	54.	87.	42.	15.	18.	35.	11.
1945	31.	24.	8.	51.	12.	34.	13.	55.	69.	9.	25.	21.
1946	14.	6.	3.	9.	19.	62.	60.	76.	36.	29.	54.	36.
1947	35.	21.	19.	29.	27.	81.	22.	48.	48.	17.	35.	22.
1948	20.	40.	16.	52.	18.	46.	78.	22.	0.	1.	21.	26.
1949	18.	8.	12.	5.	32.	39.	54.	30.	16.	35.	5.	29.
1950	30.	22.	22.	23.	21.	85.	134.	58.	20.	24.	14.	15.
1951	18.	31.	22.	22.	16.	108.	35.	80.	58.	33.	10.	23.
1952	24.	16.	8.	21.	55.	96.	50.	60.	32.	5.	20.	0.
1953	31.	17.	11.	29.	66.	105.	15.	13.	66.	1.	20.	17.
1954	33.	9.	31.	33.	53.	104.	87.	77.	86.	7.	12.	7.
1955	17.	16.	30.	56.	69.	55.	121.	6.	29.	11.	25.	27.
1956	18.	17.	7.	9.	33.	93.	43.	31.	23.	32.	7.	25.
1957	17.	4.	14.	36.	6.	51.	68.	54.	9.	27.	21.	14.
1958	5.	27.	17.	22.	16.	27.	55.	64.	25.	7.	36.	15.
1959	11.	8.	7.	29.	23.	147.	38.	24.	33.	29.	30.	13.
1960	26.	11.	5.	40.	27.	78.	43.	80.	1.	7.	25.	23.
1961	13.	27.	4.	33.	31.	39.	35.	8.	8.	25.	20.	41.
1962	25.	22.	13.	7.	36.	62.	99.	36.	22.	33.	2.	9.
1963	23.	26.	10.	11.	40.	133.	23.	38.	19.	4.	2.	19.
1964	24.	24.	11.	18.	36.	54.	18.	43.	81.	3.	21.	45.
1965	24.	24.	24.	14.	60.	105.	46.	99.	83.	2.	32.	18.
1966	24.	11.	5.	40.	20.	131.	35.	64.	5.	12.	27.	9.
1967	24.	19.	11.	29.	31.	18.	5.	24.	63.	28.	34.	20.
1968	12.	7.	4.	6.	29.	26.	29.	57.	54.	14.	6.	14.
1969	24.	12.	24.	52.	26.	29.	96.	2.	21.	64.	3.	14.
1970	33.	24.	14.	41.	24.	216.	41.	9.	32.	30.	17.	23.
1971	24.	10.	8.	15.	12.	72.	41.	6.	15.	12.	13.	14.
1972	28.	33.	9.	14.	56.	79.	34.	14.	61.	11.	11.	11.
1973	3.	12.	13.	81.	5.	31.	17.	32.	6.	6.	34.	39.
1974	33.	33.	34.	30.	100.	20.	45.	130.	18.	6.	11.	11.
1975	11.	12.	46.	50.	49.	79.	45.	50.	36.	34.	11.	21.
1976	19.	25.	53.	11.	28.	126.	50.	42.	4.	1.	5.	21.

power plant cooling water discharge. The forced evaporation losses used in carrying out the reservoir operational analysis in the SPC environmental assessment report (SPC, 1978) are provided in Table 6-6. These are basically the values computed by Spraggs adjusted to have the 1 unit scheduled maintenance outage in June instead of July and the 2 unit outages in June and August as opposed to June and July.

The MacLaren report compared the Spraggs forced evaporation results with estimates made by Shirazi. The report concluded there was no significant difference between the Spraggs and Shirazi results when Shirazi's figures were adjusted to the correct reservoir surface area, power plant size and power plant annual capacity factor. The adjusted Shirazi forced evaporation water loss for 2 unit operation was 2,834,580 m<sup>3</sup>/year (2298 ac-ft), which is 16 percent lower than the Spraggs figure of 3,369,920 m<sup>3</sup>/year (2732 ac-ft).

Shirazi provided a second more conservative estimate of total evaporation than that used in the MacLaren comparison using "higher, but realistic values of wind speed function." Comparing this with Spraggs estimate on the same basis as the MacLaren comparison shows that the more conservative estimate of 3,897,860 m<sup>3</sup>/year (3160 ac-ft) is 16 percent higher than Spraggs 3,369,920 m<sup>3</sup>/year estimate. In a check on his calculations, Spraggs also computed forced evaporation using his thermal regime predictions for the reservoir and obtained an average annual figure of 4,349,321 m<sup>3</sup>/year (3526 ac-ft). This is 29 percent higher than his 3,369,920 m<sup>3</sup>/year estimate.

Since the Spraggs forced evaporation estimate of 3,369,920 m<sup>3</sup>/year is higher than Shirazi's more liberal estimate and lower than the two conservative estimates, it is recommended that the SPC values based on Spraggs estimate and presented in Table 6-6 be adopted for reservoir operational analysis. The assumption that forced evaporation will remain fairly constant from year to year is supported by the MacLaren report conclusion that forced evaporation is insensitive to surface area and dependent mainly on the heat input rate. This is the case because increased surface area decreases excess temperatures and the effects of these two factors on forced evaporation tend to cancel each other.

TABLE 6-6

FORCED EVAPORATION LOSS FROM COOKSON RESERVOIR

Month	<u>Forced Evaporation Loss</u>	
	One Unit ( $m^3 \times 10^6$ )	Two Units ( $m^3 \times 10^6$ )
January	0.0444	0.1616
February	0.0444	0.1616
March	0.0584	0.1630
April	0.0612	0.1603
May	0.1865	0.4079
June	0.0468	0.2796
July	0.2031	0.5026
August	0.2302	0.2987
September	0.2010	0.4140
October	0.1719	0.3671
November	0.0932	0.2215
December	0.0444	0.1616
Total	1.3855	3.2995

### 7.1 Population

According to the Saskatchewan Municipal Directory, the 1977 population of the village of Coronach was 386. Using figures from the Saskatchewan Hospital Services Plan, this population has ranged from a high of 467 in 1966 to a low of 330 in 1973.

Population increase predictions were made in the Coronach Planning Study, 1976, written by the Community Planning Branch, Saskatchewan Department of Municipal Affairs. The predictions were made using Saskatchewan Power Corporation labor requirements for Stage I and II with regular multiplier effects and assuming the bulk of permanent company employees will reside in the village, the married rate for employees will be 80 percent and the average family size 3.7 persons. The report predicts a population of 850 in 1978 and stabilization of the population at 879 in 1983.

The Saskatchewan Power Corporation has since altered their labor predictions and has subsequently predicted the population of Coronach will stabilize at approximately 1,100 persons. Keith Consulting has used a design population of 1,200 for the design of water treatment, distribution, and wastewater handling facilities.

### 7.2 Water Supply

The present water source for the village is a 15 m (50 ft) deep, 20 cm (8 in) diameter well rated at 4.2 l/s (55 Igpm) adjacent to the PFRA reservoir on Girard Creek. The well is identified as point number 1 on Figure 7-1. Water treatment includes aeration, filtration for iron removal, and chlorination. A 340 m<sup>3</sup> (75,000 Imp gal) capacity standpipe south of the village provides storage.

The village has recently developed a new 14 m (47 ft) deep well (number 2 in Figure 7-1). A new 500 m<sup>3</sup> (110,000 Imp. gal) treated-water storage reservoir and a new treatment plant which includes potassium permanganate-filtration for iron and manganese removal and gas chlorination was scheduled for completion during 1978.

The new treatment plant and mains were designed for a population of 1,200 requiring an average flow of 545 m<sup>3</sup>/day (120,000 Igpd or 160 ac-ft/year). Preliminary investigations have been completed for a third well (number 3 in Figure 7-1), located 370 m east of Well No. 2. This well will be developed when required.

### 7.3 Wastewater Treatment

#### (a) Present

Municipal wastewater from the village is pumped to a two-cell lagoon shown in Figure 7-1. The lagoon is discharged twice yearly into a small tributary of Girard Creek.



Figure 7-1. Municipal water and waste handling facilities at Coronach.

In 1975, the village was advised by the Saskatchewan Department of the Environment that the amount of nutrients (particularly phosphorus) discharged into Girard Creek must be reduced. The lagoon was treated with alum to remove phosphorus before discharging in June and October of 1977.

(b)      Future

Using a design population of 1,200, a new sewage treatment and effluent irrigation system is being designed for Coronach. A two cell lagoon system (first cell, 2.75 ha, second cell, 5.63 ha) will be constructed as shown in Figure 7-1. The second cell (holding cell) is designed to provide 220 days of storage for the wastewater. The effluent from the second cell will be used to irrigate forage crops. Preliminary geotechnical investigations in this quarter section have shown that the area is suitable for effluent irrigation. Further geotechnical work is being conducted. Contamination of groundwater is not expected to be a problem.

The village is currently negotiating for purchase of the land and construction of the new lagoon system is expected to be completed during 1979.

7.4      Solid Wastes

The old nuisance grounds has been leveled and buried with the required 0.6 m (2 ft) of soil. The water table at the site is 15 m (50 ft) below the ground surface in till.

A new 4 ha (10 ac) site has been developed for a modified landfill system as shown in Figure 7-1. The site will be operated in accordance with Saskatchewan Regulation 198/72. The regulation requires that solid wastes be compacted and covered with at least 0.6 m (2 ft) of soil, once per month from May to October. The site must be 460 m (1,500 ft) from a water body, water course, or dwelling, and 90 m (300 ft) from a highway, street, road, or railway.

The site has been approved by the Saskatchewan Department of the Environment with the following conditions:

1. Periodic burning of dry material (trees, lumber, etc.) under controlled conditions and placement in a dry disposal area is acceptable. This dry disposal area should not be a trench or hole of any kind. The burning of tires or waste oil is not permitted.
2. The bottom of the landfill cells shall be at least 1.5 m (5 ft) above the groundwater table.
3. Liquid wastes shall not be deposited at the landfill site.

Some geotechnical investigations in the area have indicated that the groundwater table is at least 15 m (50 ft) below the surface.

7.5

Drainage

There is no storm sewer system in Coronach nor is a system planned for the proposed subdivision. Therefore, runoff occurs naturally through street drainage. Storm runoff enters tributaries of Girard Creek east and west of the village. These tributaries drain into the PFRA reservoir. Because of the absence of industries in Coronach and because of the retention time in the PFRA reservoir, the storm drainage from the village is not expected to significantly affect Cookson Reservoir.

7.6

Emissions

Since there are no industries in Coronach, the most significant air emissions in the village will be from automobile exhausts and from home heating systems. These emissions should not affect water quality in Cookson Reservoir.

### 8.1 Suggested Quality Criteria

Although they had not been finalized at the time the following assessment of mitigation requirements was conducted, the Uses and Water Quality Objectives Committee provided the Plant, Mine and Reservoir Operations Committee with the following tentative water quality objectives for the International Boundary:

#### Total Dissolved Solids (TDS)

- (i) the long-term (10-year) arithmetic mean of the monthly flow-weighted means during the irrigation period (May 1 to September 30) should not be caused to exceed 1000 mg/l, and
- (ii) the short-term (any 3-consecutive months) arithmetic mean of the monthly flow-weighted mean during the irrigation period (May 1 to September 30) should not be caused to exceed 1500 mg/l.

#### Boron (B)

- (i) the long-term (10-year) arithmetic mean of the monthly flow-weighted means during the irrigation period (May 1 to September 30) should not be caused to exceed 5.0 mg/l, and
- (ii) the short-term (any 2-consecutive months) arithmetic mean of the monthly flow-weighted mean during the irrigation period (May 1 to September 30) should not be caused to exceed 8.0 mg/l.

#### Trace Metals

<u>Metal</u>	<u>Dissolved (mg/l)</u>	<u>Total (mg/l)</u>
Aluminum (Al)	0.1	
Cadmium (Cd)		0.0012
Chromium (Cr)		0.05
Copper (Cu)	0.005	1.0
Flouride (F)		1.5
Iron (Fe)		0.5
Lead (Pb)		0.03
Manganese (Mn)		0.3
Mercury (Hg)	0.002	
Zinc (Zn)		0.3

#### Other

Sulphate ( $\text{SO}_4$ )	< 800 mg/l
Nitrate ( $\text{NO}_3$ )	< 10 mg/l
Ammonia ( $\text{NH}_3$ )	< 0.02 mg/l
Sodium Absorption Ratio (SAR)	< 10 mg/l

The more detailed requirements for TDS and B are a result of the link between these constituents and irrigation needs and the early indication through ash leachate analyses and reservoir computer models runs that these constituents would likely require mitigation. The Uses and Water Quality Objectives Committee suggested that Cd and un-ionized ammonia ( $\text{NH}_3$ ) contributions from the plant and mine operation be closely examined. It was considered that all other parameters would be within the requirements of the recommended border objectives.

In addition to the above objectives recommended by the Uses and Water Quality Committee, other committees defined possible mitigation needs or made reference to potential problem elements during the course of their studies or in draft submissions to the Board. These included:

- (i) a request by the Biology Committee that the possibility of releasing a  $20 \text{ m}^3/\text{s}$  continuous flow for 2 consecutive days each year be considered. This release was judged necessary for maintenance of habitat for fish and ducks and spawning areas downstream.
- (ii) an indication in the Groundwater Quantity and Quality Committee's draft report that Mo, Cr,  $\text{NO}_3$  and B would be the most likely elements of concern in ash lagoon seepage to groundwater
- (iii) indications in the foregoing sections of this report of potential mitigation requirements for the mining, power plant and reservoir operation.

Each of the concern areas identified above has been examined by the Plant, Mine and Reservoir Operations Committee in assessing mitigation options for Poplar River.

## 8.2 Evaluation of Proposed Operation

### 8.2.1 Description of Reservoir Operation Model

A reservoir operation computer model was used by the Plant, Mine and Reservoir Operations Committee to assist in the provision of information to other committees, better identify those areas that might require mitigation and test the sensitivity of various mitigation options. The model is a revised version of the simulation model developed and used by Montreal Engineering Company Ltd. in studies conducted for the Poplar River environmental assessment (SPC, 1978). A full description of the model is contained in Annex J. The model is designed to compute reservoir water quantity and quality on a monthly bases for any designated time period.

For the Poplar River study, power plant operation with the 46 years of hydrologic and meteorologic conditions from 1931-76 were simulated. The model can be run with any of the no power plant, 1 unit or 2 unit options, and with or without lime softening of the reservoir water.

The model considers each of the following factors in the simulation of monthly reservoir operation:

- runoff from the East Poplar River basin
- concentrations of various elements in the natural flows to Cookson Reservoir
- natural and forced evaporation from the reservoir and resulting concentration of elements
- natural evaporation from the ash lagoons and resulting concentration of elements in the lagoons
- direct contribution of precipitation to the reservoir and ash lagoons
- quantity and quality of mine dewatering additions to the reservoir
- power station and ash lagoon effluents and direct chemical additions
- lime and acid additions for reservoir water softening and the chemical balance adjustments resulting in the reservoir
- downstream riparian, release-on-demand and spillage flows.

The quantity and quality of the natural and wastestream inputs to the reservoir have been described in the foregoing sections of this report.

To enable the model to more accurately reflect certain details of power plant operation and provide output data tailored to the needs of the current study, the Plant, Mine and Reservoir Operations Committee made the following revisions to the original MECO model:

- (i) Encorporation of a routine for computing flows and quality at the International Boundary based on the various contributing flows including groundwater baseflow, seepage beneath and around Morrison Dam, seepage from the ash lagoons and releases through the outlet works.
- (ii) Major revisions to the ash lagoon operational procedure and flow rates.

(iii) Re-routing of contributing effluent streams from the plant to the reservoir or ash lagoon as appropriate.

(iv) Adding an SAR computation.

(v) Altering and expanding the model output reports to include quantities and qualities from the ash lagoon, in the reservoir and at the International Boundary and specific statistics such as May to September volume weighted means and frequency of occurrence of concentration levels.

The original MECO model and a version with most of the subsequent revisions was provided to the Surface Water Quality Committee for their use in assessing the effects of the mining, power plant and reservoir operation.

#### 8.2.2 Components of International Boundary Flow

Most aspects of assumptions made regarding components of flow contributing to discharge at the International Boundary were discussed in Section 6. However, it may be helpful to summarize those assumptions here.

The discharge ( $Q$ ) at the boundary was considered to be made up of the following 5 flow components:

<u>Flow Component</u>	<u>Flow 1/s (cfs)</u>	<u>Assumed Worst Case Water Quality</u>
1. Post-reservoir baseflow ( $Q_1$ )	0.5x pre-reservoir baseflow	Pre-reservoir baseflow quality (B)
2. Seepage under Morrison Dam ( $Q_2$ )	25.5 (0.9)	Reservoir water quality (R)
3. Seepage from reservoir around Morrison Dam ( $Q_3$ )	7.1 (0.25)	Reservoir water quality (R)
4. Seepage from ash lagoons ( $Q_4$ )	Variable	Computed monthly ash lagoon water quality (A)
5. Low-flow release at Morrison Dam ( $Q_5$ )	$Q_5 = Q - (Q_1 + Q_2 + Q_3 + Q_4)$	Reservoir water quality (R)

After computing the actual monthly rates for  $Q$ ,  $Q_1$ ,  $Q_4$  and  $Q_5$ , the model then computes the quality (I) of the total flow ( $Q$ ) crossing the International Boundary from:

$$I = [B(Q_1) + R(Q_2 + Q_3 + Q_5) + A(Q_4)]/Q$$

### 8.2.3 Modelling Assumptions for Testing Sensitivity

#### (a) Ash Lagoon Leachate and Seepage Rates

In addition to the estimated average rate of leaching of material from the ash, high and low leaching rates were also selected to determine the sensitivity of reservoir and border water quality to dissolution of elements from the ash. Based on the range of values obtained in the various static ash leaching tests, the following levels were selected for the sensitivity analysis:

<u>Element</u>	<u>Added concentration in Ash Lagoon (mg/l)</u>		
	<u>Low Estimate</u>	<u>Average Estimate</u>	<u>High Estimate</u>
Na	20	30	65
K	1	5	10
Ca	0	0	100
Mg	0	0	10
B	6	10	15
HCO <sub>3</sub>	-200	-100	0
CO <sub>3</sub>	30	40	55
SO <sub>4</sub>	100	180	240
NO <sub>3</sub>	0	1	2
Cl	3	5	20
TDS (computed)	-4	160	445

It should be noted that the "average estimate" of HCO<sub>3</sub> reduction of 100 mg/l is considerably lower than the representative value of 200 mg/l determined from the ash leachate tests and listed in Table 5-7. Since the "average estimate" will be considered most representative of expected conditions under actual operation, the lower value for HCO<sub>3</sub> reduction was chosen to ensure that TDS contributions from the ash lagoons are not underestimated. Consideration was given to observed HCO<sub>3</sub> reduction in the ash lagoons at the Boundary Dam G.S., where the average reduction is less than 100 mg/l, and the variability in intake water quality expected.

With the proposed once-through lagoon system, the seepage rate to the reservoir is too small in proportion to the decant flow to make any significant difference. However, seepage to the East Poplar River downstream of Morrison Dam must be considered. In analyzing the proposed operation, an average and a high seepage rate to the East Poplar River are assumed. These are 2.0 l/s (0.07 cfs) and 17.0 l/s (0.6 cfs) respectively per lagoon cell in operation.

In the reservoir operation model computation procedure for lagoon operation, more than three lagoon cells can theoretically be operating in series at any one time. The simultaneous use of four lagoon cells in series was observed in some runs. This occurs because the model does not compute partial month values for filling, settling and decant. In actual practice, sequential operation with three lagoons should be quite adequate. Any errors attributable to the full month computation procedure are likely to have an insignificant effect on reservoir and border water quality.

(b) Mine Dewatering Rate

Because there is some uncertainty as to what the mine dewatering rate over time will be in actual practice, two rates are tested in the model. One is a rate decaying, as described in Section 3.6.1, from 131 l/s (4.6 cfs) at the start of the simulation to 43 l/s (1.5 cfs) after 8 years of plant operation and remaining constant thereafter. The procedure used in the original model of adjusting dewatering rates annually between 30 l/s and 60 l/s after the first 8 year period is retained in the model as adequately reflective of the steady state dewatering rate.

The other dewatering rate used is a continuous rate of 100 l/s based on information provided by the Groundwater Quantity and Quality Committee on expected pumping rates and surface water depletion over time. It is assumed that the difference between the estimated pumping rates and estimated surface water depletion in Girard Creek and the East Poplar River upstream of Cookson Reservoir as a result of the dewatering would constitute the expected net increase in water available to the reservoir. The net increase in inflow computed in this way for successive 5 year periods of power plant operation are:

<u>Period</u>	<u>No. of Years</u>	<u>Reservoir Inflow (l/s)</u>
1979-1983	5	18.6
1984-1988	10	82.8
1989-1993	15	23.7
1994-1998	20	68.3
1999-2003	25	157.4
2004-2008	30	113.4
2009-2013	35	144.7
2014-2018	40	-38.7
2019-2023	45	-17.9

Since in these estimates pumping is assumed to cease after 35 years, the two negative values are not indicative of plant operating conditions. It can be seen from the remaining values that a net inflow increase greater than 100 l/s could occur during some periods.

A high rate of net inflow increase due to dewatering of 100 l/s continuous was selected for examining the effects of increased pumping. This rate means there is substantially more water available to the reservoir than with the previous assumption and whether this will improve or deteriorate reservoir water quality will depend on what the actual quality level is in the reservoir at any given time.

(c) Evaporation Loss

Since differences did exist in the various estimates of both natural and forced evaporation, the sensitivity of an underestimation was assessed by running the model with a forced evaporation rate 20 percent higher than the best estimate figures provided in Section 6.

8.2.4 Relative Significance of Contributing Effluent Sources

To provide a general appreciation of which effluent sources are contributing the largest loading of elements to the reservoir and hence which mitigation measures would be most effective in improving water quality, the TDS and B additions from the major sources were compared. The results of the comparison are contained in Table 8-1. The loadings from the mine dewatering water were computed from the difference in concentrations between mine dewater and assumed reservoir water concentrations of 700 mg/l for TDS and 2 mg/l for B.

It can be seen from the table that the ash lagoon decant is by far the greatest contributor to both TDS and B loadings to the reservoir and hence the area where mitigation might be most effective.

8.2.5 Results of Model Runs

Several computer runs made by the Plant, Mine and Reservoir Operations Committee were designed to determine the effects of the 1 and 2 unit operation with the range of ash lagoon leachate and seepage, dewatering and forced evaporation rates described above. The results of these runs are listed in Table 8-2 along with the results of additional runs made to test the effects of using reservoir water softening to control condenser tube scaling and the effects of a higher rate of forced evaporation from the reservoir. It should be noted that the concentration of elements in inflows to the reservoir used in all runs up to Run No. 8 were those used in the environmental assessment report (SPC, 1978). In all subsequent runs, the concentration versus monthly flow relationships developed by the Surface Water Quality Committee were used to determine inflow concentrations. Run Nos. 9 and 10b are actually re-runs of earlier Run Nos. 1 and 3 to check the effect of the refined inflow concentrations. It was concluded from these comparisons that the difference in the results obtained with the revised concentrations was not sufficient to warrant re-running all earlier runs and that all concentrations listed in Table 8-2 are adequately reflective of the magnitudes of border water quality that would occur with the proposed system. The average seepage rate from the ash lagoons was also revised for Run No. 10b to 2.0 l/s per lagoon from

TABLE 8-1 COMPARISON OF LOADINGS FROM DIFFERENT EFFLUENT SOURCES

EFFLUENT SOURCE	AVG. ANNUAL FLOW (1/s)	CONCENTRATION ADDITIONS			CONTRIBUTION TO RESERVOIR <sup>1</sup>		
		Fe (mg/l)	B (mg/l)	TDS (mg/l)	Fe (Kg/yr)	B (Kg/yr)	TDS (Kg/yr x 10 <sup>6</sup> )
<b>1 UNIT</b>							
Mine Dewatering							
- low rate	43	(2.0) <sup>2</sup>	- 0.3	400	(2,710)	- 410	0.54
- high rate	100	(2.0)	- 0.3	400	(6,310)	- 950	1.26
Ash Lagoon Decant							
- low leaching rate	150	0.0	6.0	0	0	28,400	0
- avg. leaching rate	150	0.0	10.0	160	0	47,340	0.76
- high leaching rate	150	0.0	15.0	440	0	71,000	2.08
Ash Lagoon Seepage - Once Thru	4	0.0	10.0	160	0	1,260	0.02
Ash Lagoon Seepage - Recirc.	1.5	(1.0)	48.0	3,850	(47)	2,270	0.18
Fireside and Preheater Washing	0.25	1,500	(50.0)	10,000	11,830	(390)	0.08
Equiv. Contribution from Net Nat. Evap.				25,000 <sup>3</sup>		8.23	
Equiv. Contribution from Forced Evap.				5,000		1.65	
<b>2 UNITS</b>							
Mine Dewatering							
- low rate	43	(2.0)	- 0.3	400	(2,710)	- 410	0.54
- high rate	100	(2.0)	- 0.3	400	(6,310)	- 950	1.26
Ash Lagoon Decant							
- low leaching rate	309	0.0	6.0	0	0	58,510	0
- avg. leaching rate	309	0.0	10.0	160	0	97,510	1.56
- high leaching rate	309	0.0	15.0	440	0	146,270	4.29
Ash Lagoon Seepage - Once Thru	6	0.0	10.0	160	0	1,890	0.03
Ash Lagoon Seepage - Recirc.	1.5	(1.0)	48.0	3,850	(47)	2,270	0.18
Fireside and Preheater Washing	0.5	1,500	(50.0)	10,000	23,670	790	0.16
Equiv. Contribution from Net Nat. Evap.				25,000		8.23	
Equiv. Contribution from Forced Evap.				12,000		3.95	

<sup>1</sup> Kg/yr = (1/s) x (mg/l) x 31.5576<sup>2</sup> Bracketed ( ) numbers are assumed based on limited information.<sup>3</sup> Contributions from concentration due to evaporation are based on a full reservoir and initial TDS and B levels of 1000 mg/l and 3 mg/l respectively.

TABLE 8-2 RESULTS OF RESERVOIR OPERATION MODEL RUNS FOR PROPOSED SYSTEM

COMPUTER MODEL RUN		WATER QUALITY AT INTERNATIONAL BOUNDARY								IDENTIFICATION OF CONCENTRATION LISTED	
NO.	DESCRIPTION	Na/K (mg/l)	Ca (mg/l)	Kg (mg/l)	T. Alk. (mg/l)	SO <sub>4</sub> (mg/l)	Cl (mg/l)	TDS (mg/l)	SAR	B (mg/l)	
9	No units, no mine dewatering (reservoir evaporation only)	221 (123) 201 (132)	200 (132) 165 (135)	252 164 232 174	617 401 569 422	285 164 264 176	9.6 5.0 8.2 5.2	1010 626 937 662	4.7 3.0 4.4 3.2	2.1 1.2 1.9 1.3	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.
6	1 unit, avg. leachate rate, avg. seepage (3.5 l/s per ash lagoon)	355 (290) (330) (210)	283 (180) (246) (180)	336 (220) (321) (230)	762 (500) (711) (500)	626 318 494 321	19.9 (9.0) (18.0) (9.0)	1739 974 1385 977	6.8 4.5 5.7 4.4	12.7 4.7 8.3 5.0	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.
10b*	2 units, avg. leachate rate, avg. seepage (2.0 l/s per ash lagoon)	506 (183) 364 (183)	383 (176) 284 176	445 206 347 206	796 426 663 435	1143 354 753 354	46.5 13.2 29.5 13.2	2411 926 1795 953	7.6 4.0 6.4 4.1	31.4 8.1 18.7 8.1	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.
2	2 units, high leachate rate, avg. seepage (3.5 l/s per ash lagoon)	713 (250) (572) (275)	713 (250) (545) (275)	540 (230) (457) (240)	1174 (550) (989) (550)	1456 450 991 461	88.7 (20.0) (66.8) (25.0)	3378 1240 2413 1259	8.7 5.0 7.3 5.0	45.4 11.3 28.9 11.9	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.
5	2 units, low leachate rate, avg. seepage (3.5 l/s per ash lagoon)	562 (210) (464) (220)	434 (190) (345) (190)	512 (230) (438) (240)	669 (450) (626) (425)	1015 351 718 355	41.3 (12.0) (32.8) (13.0)	2317 996 1753 998	7.9 4.7 6.8 4.6	20.3 5.5 13.2 5.7	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.
4	2 units, avg. leachate rate, high seepage (17.0 l/s per ash lagoon)	636 (225) (538) (225)	462 (180) (390) (180)	573 (225) (485) (225)	1030 (450) (873) (450)	1360 402 958 413	51.6 (13.0) (42.5) (13.0)	2965 1052 2170 1059	8.6 4.7 7.2 4.7	33.8 8.5 22.6 9.2	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.
8	2 units, avg. leachate rate, avg. seepage (3.5 l/s per ash lagoon), reservoir water softened	587 (200) (482) (225)	195 (132) (152) (129)	404 (220) (361) (220)	530 (440) (483) (420)	1217 399 845 407	46.9 (13.0) (36.8) (14.0)	2296 1021 1739 1022	11.1 5.3 8.8 5.3	31.4 8.1 20.2 8.5	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.
16	2 units, avg. leachate rate, avg. seepage (3.5 l/s per ash lagoon), forced evap. increased by 20 percent	782 (200) (578) (200)	584 (180) (433) (190)	681 (225) (522) (225)	1136 (475) (908) (475)	1834 (350) (1293) (400)	73.9 (15.0) (52.2) (15.0)	3756 (1020) 2660 (1090)	9.5 (4.0) (8.1) (4.2)	51.4 (10.0) 32.2 (10.8)	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.
10c	2 units; avg. leachate rate; avg. seepage (2.0 l/s per ash lagoon); mine dewatering continuous at 100 l/s	412 193 326 200	347 193 281 196	381 212 319 220	798 471 690 487	806 250 605 362	28.7 11.8 20.7 12.1	1958 967 1555 903	6.6 4.1 5.7 4.2	13.0 6.9 12.4 7.1	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.

NOTE: Concentrations in reservoir inflows were from SPC environmental assessment report for Run Nos. 1-8. Concentrations in reservoir inflows were from Surface Water Quality Committee regression equations for runs after Run No. 8.

All concentrations listed except the max/min monthly values are flow weighted.

All bracketed figures are approximated from earlier output listings which did not provide data in the exact form required.

\* Run No. 10b is considered most representative of expected operating conditions.

the earlier 3.5 l/s rate. Hence, caution should be exercised in making exact comparisons between runs.

Run No. 10b is considered to be most representative of expected operating conditions, with the average rate of leaching uptake in the ash lagoons and the average seepage rate to the East Poplar River. Table 8-2 shows that the long-term May to September mean TDS level is less than the 1000 mg/l limit by about 47 mg/l or 5 percent. The long-term May to September mean B level exceeds the 5 mg/l limit by 3.1 mg/l or 60 percent. The more detailed computer output listings for Run No. 10b show the number of 3 month or 2 month periods in 46 years that the short-term irrigation season limits for TDS and B respectively are exceeded. These are:

Number of 3-consecutive month periods with TDS	1500 mg/l	7
Number of 2-consecutive month periods with B	8 mg/l	73

These results indicate that remedial action will be needed to reduce both TDS and B levels in the flow crossing the border. Although Run No. 10b is considered most indicative of the expected 2 unit power plant operation, each of the other runs listed in Table 8-2 represent conditions that could occur with the proposed waste management system, given the possible range of variation in model input parameters. The following are observations and comments on the results of these runs.

- (i) The long-term flow-weighted May to September mean TDS equals or exceeds the 10-year limit of 1000 mg/l for all conditions of 2 unit operation examined except those of Run No. 10b where a lower average ash lagoon seepage rate was used than in most other runs. A mean May to September TDS of 1000 mg/l infers that a number of 10-year May to September means in the 46-year simulation period are greater than 1000 mg/l. The maximum computed May to September TDS is also greater than the 3 consecutive month limit of 1500 mg/l for all 2 unit runs.
- (ii) Both the short-term objective for B of 8 mg/l and the long-term objective of 5 mg/l are exceeded for all 2 unit runs with the excess being substantial in most cases.
- (iii) The results of Run No. 5 indicates that both the long-term and the single season levels for TDS and B would be considerably reduced if the rate of dissolution of elements from the ash was lower than expected. However, even with the low leachate rate, the single season B level of 8.0 mg/l is exceeded for 27 periods of 2 consecutive months during the irrigation seasons in the 46 years analyzed. There is some evidence suggesting the leachate rates could be less than the average rates used for Run No. 10b but no strong evidence suggesting they will be as low as the rates used in Run No. 5. The lower leachate rate does not lower the border TDS levels significantly.

- (iv) The difference in TDS and B levels between Run No. 10b and Run No. 4 are not as much as expected suggesting that with the proposed once-through ash lagoon system the decant to the reservoir has much more influence on border water quality than does the seepage rate. To reduce TDS and B levels at the border, measures to reduce the contribution to Cookson Reservoir from the decant water will likely be most effective. However, if the decant water contribution is reduced, the influence of the seepage rate can be expected to increase suggesting that it be limited to rates of 2.0 l/s per lagoon or less.
- (v) The results of Run No. 8 suggest that lime softening of the reservoir water as a scale control measure would reduce the maximum TDS levels at the border somewhat by reducing the total alkalinity but would have little effect on average TDS levels because of an increase in  $\text{SO}_4$ . This balance between reduction in total alkalinity and increase in  $\text{SO}_4$  may change if average TDS levels are lower, and softening may be an effective means of TDS reduction. Boron concentration, however, would not be affected by softening.
- (vi) Run No. 16 suggests that if evaporation loss was higher than estimated, significant increases in both TDS and B would occur at the border. The affects of possible error in the evaporation estimates should be re-assessed in the selection of mitigation alternatives.
- (vii) The effect of increased evaporative loss on the level of Cookson Reservoir was also examined. The higher evaporation rate in Run No. 16 reduces the minimum monthly water level for the period to 744.73 m as compared to a minimum level of 745.42 in Run No. 10b. This difference is fairly small but since the levels are near the limit for power plant operation, the ability to maintain full operation during an extended dry period may be questionable if evaporation is higher than expected.
- (viii) The results obtained from Run No. 10c demonstrate the effect on border water quality of a continuous net supply from mine dewatering of 100 l/s. Comparing the Run No. 10c results with Run No. 10b shows that the mean May to September TDS and B levels of 953 mg/l and 8.1 mg/l of Run No. 10b became 998 mg/l and 7.1 mg/l respectively indicating a deterioration in TDS and an improvement in B concentration. The improvement in B levels would, however, be less pronounced if reservoir water quality were better on average and deterioration in TDS would be more pronounced. Since mitigation will be required to improve reservoir water quality, the effect of increased dewatering should be re-examined in conjunction with the proposed mitigation option. The affect of the higher dewatering rate on reservoir levels observed from Run No. 10c results would hold for most mitigation options. The

minimum reservoir level observed with the higher rate is 748.55 m as compared to 745.42 for Run No. 10b., which is a substantial improvement.

Reservoir levels have been plotted in Figure 8-1 for the no power plant, 1 unit and 2 unit operating conditions. The graph shows the pattern of levels that could be expected over the 46 year study period.

#### 8.2.6 Cost of Proposed System

Estimated capital and annual costs for the proposed systems at the Poplar River power development are required for use in evaluating the significance of the additional costs attributable to various mitigation alternatives. The most valid basis for comparison is percent increase in annual cost or cost of energy produced.

It should be emphasized that all costs provided in this report are of necessity only very rough estimates and subject to change based on actual design of the facility being costed.

The latest (February, 1979) figures available to the Plat, Mine and Reservoir Operations Committee on estimated captial cost (in 1979 dollars) of the Poplar River power plant are:

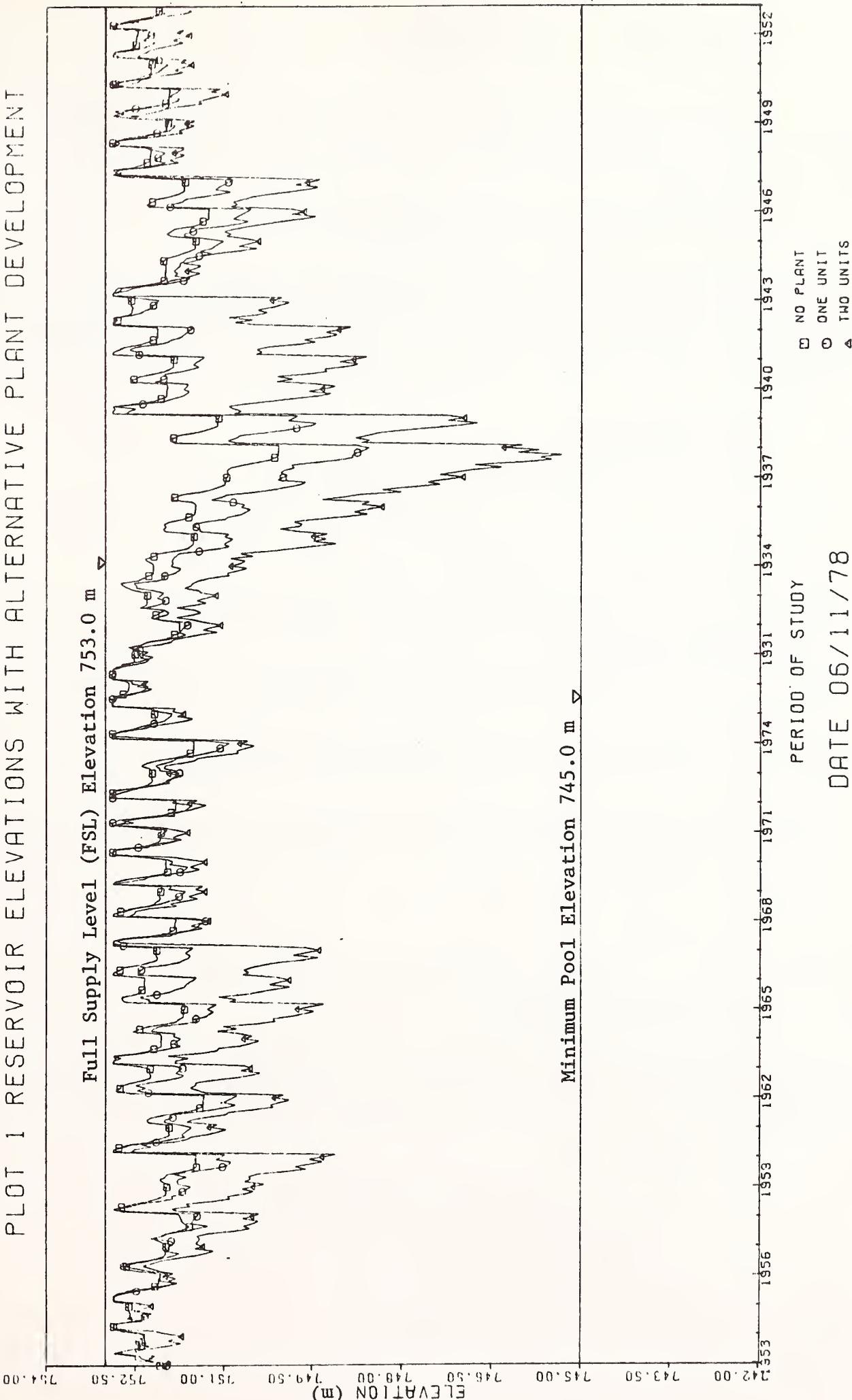
1st unit	\$192,000,000
2nd unit	<u>\$162,000,000</u>
Total 2 units	\$354,000,000

The total annual cost includes interest and amortization, the cost of supplying the lignite fuel, operation and maintenance, interim replacement, insurance, taxes administraion and other annual expense items. These are included in the projected 1.69 cent/kwh cost of producing electrical energy at Poplar River. A 1.69 cent/kwh production cost translates into a total annual cost for 2 units operating ( $3.924 \times 10^9$  kwh/year) of \$66,300,000.

Since a number of the potential mitigation measures are alternative ash disposal methods, it is also useful to have a base cost estimate for the proposed system. The proposed once-through ash disposal system costs, including the cost of the electrostatic precipitator and the proposed method of lagoon lining by reworking the upper till layer, are estimated to be:

FIGURE 1-1 PLOT OF MONTHLY RESERVOIR LEVELS - 46 YEARS

PLOT 1 RESERVOIR ELEVATIONS WITH ALTERNATIVE PLANT DEVELOPMENT



Capital Cost	\$13,800,900
Annual Cost	\$ 2,043,770

Mitigation costs discussed in the following sections will be compared to both the estimated total annual cost for the project of \$66,300,000 or the estimated annual cost of the proposed ash disposal system of \$2,043,770.

### 8.3 Identified Mitigation Alternatives

A whole matrix of mitigation alternatives are possible to accomplish a reduction in International Boundary levels for TDS and B. The mitigation possibilities for these as well as other potential pollutants are listed below and selected options are described more fully in Sections 8.4 and 8.5.

#### 8.3.1 Mining Operations

- (i) treat mine dewatering discharge (TDS, B)
- (ii) treat mine pit water discharge (TDS, B)
- (iii) treat diverted surface water (TDS, B)
- (iv) divert mine dewatering discharge to East Poplar River below Morrison Dam
- (v) divert all mine effluents to East Poplar River below Morrison Dam
- (vi) treat all effluents and divert to East Poplar River below the dam
- (vii) control effluents for optional release either to the reservoir or below the dam
- (viii) increase the mine dewatering rate
- (ix) utilize favourable reclamation and runoff control options.

#### 8.3.2 Transportation Systems

Since no significant affect on water quality is predicted, no mitigation measures have been identified for the project transportation systems except for certain operating procedures described in Section 4.1.

#### 8.3.3 Ash Disposal System

The major ash disposal system alternatives are:

- (i) Combined ash, once-through lagoon (proposed system)
- (ii) Combined ash, recirculating lagoon.
- (iii) Dry flyash, once-through bottom ash lagoon.
- (iv) Dry flyash, recirculating bottom ash lagoon.
- (v) Dry flyash, dewatered bottom ash, mechanical ash transport to landfill or mine.

Although systems (ii) through (v) are major mitigation measures in themselves, more specific mitigation features can be considered with each option, including the first. With each of the systems employing wet ash transport to a lagoon, the lagoon could:

- be lined, with or without under drainage
- have a downstream grout curtain
- have pumpwells downstream
- have treated decant or blowdown
- be constructed in a new location (some or all lagoons)
- have dust control measures for exposed dry ash areas
- be reclaimed for productive use
- have a limited number of cells with periodic ash removal to a landfill or to the mine

If system (v) was employed, the selected landfill site could:

- be totally above water table
- be lined
- have runoff containment and treatment
- be located remote from the station
- be stabilized and vegetated as sections are completed
- have special dust control measures for ash transport and deposit.

#### 8.3.4 Other Power Plant Wastewaters

- (i) Line the sewage lagoon, evaporation pond and holding pond.
- (ii) Transport reverse osmosis rejects and washwaters from the coal gallery, air preheater and boiler fireside washing to the evaporation pond or holding pond.
- (iii) Enlarge the evaporation pond and holding pond or construct new ponds as required for a waste management system designed to accomodate a new ash disposal procedure.
- (iv) Enlarge and/or move the sewage irrigation area.
- (v) Treat releases from the evaporation pond and holding pond.
- (vi) Provide improved containment for transformer oil leaks.

#### 8.3.5 Power Plant Cooling System

- (i) Use mechanical condenser cleaning.
- (ii) Increase pumping capability.
- (iii) Reverse intake and outfall locations.
- (iv) Operate all pumps when spills less than the cooling water flow rate occur at Morrison Dam.
- (v) Install fish bypass facilities at the intake.
- (vi) Install barrier to prevent fish from residing in outfall channel.

#### 8.3.6 Reservoir Operation

- (i) Treat riparian flow release.
- (ii) Collect and treat low flows (including ash lagoon seepage) and treat or pump back to reservoir and replace with reservoir water.
- (iii) Contain the release-on-demand volume annually with an overflow weir upstream of Cookson Reservoir and release by gravity flow pipe below Morrison Dam when required.
- (iv) Lime soften the reservoir water.
- (v) Dilute reservoir water or reservoir releases with water diverted from the Poplar River.

### **8.3.7      Village of Coronach**

With the proposed waste disposal facilities for the Village of Coronach, no significant affects on water quality in Cookson Reservoir or at the International Boundary are anticipated. Therefore, no further mitigation measures have been identified.

### **8.3.8      Other Alternatives**

- (i) Prevent the possibility of future overflows from Fife Lake.
- (ii) Dewater other areas in the upper East Poplar River basin to provide increased net inflow to Cookson Reservoir.
- (iii) Treat flow crossing the International Boundary.
- (iv) Have longer forced outage of one or both power plant units during summer.

## **8.4            Evaluation of Selected Mitigation Alternatives**

### **8.4.1        Mining Operation Alternatives**

A number of practices and procedures have been assumed in preparing estimates of precipitation runoff and sediment yield from the mining operation. Employment of these practices would assure minimizing adverse effects to the mining activities; these assumptions have been stated in Sections 3.6.3 and 3.6.4. The Plant, Mine and Reservoir Operations Committee recommends the adoption of these practices and design considerations by the Saskatchewan Power Corporation to help minimize potential adverse effects of the project.

Nine potential mitigation options were identified in the area of mine operations. Each option was examined to determine what improvement in water quality would result from its adoption, how reliably an option would function, and what would be the relative cost and effectiveness of operating each option.

#### **(i)        Proposed treatment of dewatering discharge to reduce boron and total dissolved solids.**

A system of collection pipes and a water treatment facility could be installed to reduce the TDS and B content of water pumped from the overburden during dewatering operations preparatory to mining. Such treatment could conceivably reduce the TDS in dewatering effluent to approximately 100 mg/l and boron levels to less than 1 mg/l. A facility capable of treating up to 100 l/s could affect up to 8 percent of the cooling water reservoir volume annually and could improve water quality in the reservoir and in the East Poplar River, especially during prolonged dry periods. However, the cost of installing and operating such a facility and the requirement to contain and dispose of back-flushing effluents and cleaning chemicals argue against the adoption of this alternative.

(ii) Treatment of mine sump discharge to reduce boron and total dissolved solids.

This option is similar to option (i) in the treatment proposed, except that treated volumes of water would be much lower and a system of collection pipes would not be required. Because the anticipated mine sump water quality is similar to observed pumpwell water quality, and because the volume of mine sump discharge is relatively small (less than 10 percent of dewatering volume), this option would not offer any significant improvements in overall water quality. In addition, the use of this option would entail a relatively large capital expense.

(iii) Treatment to reduce total dissolved solids and boron content of surface waters diverted around the mine site.

This option, though possible, does not appear practical. No continuous flow in the diversion system is expected. Thus, a treatment plant would have to be large enough to handle runoff in spring and from rainfall events, but would remain idle during most of the year.

Improvements in water quality would be marginal because runoff is expected to be of generally good quality, and the treated water would be discharged at a time when water quality problems are minimal.

(iv) Diversion of mine dewatering discharge into the East Poplar River below Morrison Dam.

Mine dewatering flows could be collected and transported by pipeline to an outfall on the East Poplar River downstream of Cookson Reservoir. Dewatering flows are expected to range from 40 to 115 l/s and will contain about 1.7 mg/l of B and more than 1,000 mg/l of TDS. Hence, if combined with reservoir outlet flows they could lower the B concentrations at the border but would not help achieve the recommended TDS objectives.

The length of pipeline required would be approximately 14.5 km; the collection network would add about 5 km of additional pipe to the system and the pipeline would have to be buried below the frost line. Operating costs and installation costs would be moderate.

(v) Diversion of all mine effluent water into the East Poplar River downstream from Cookson Reservoir.

With this option, mine sump drainage, dewatering flows, and diverted runoff from intercepted drainages would be collected and diverted into the East Poplar River below Morrison Dam. Although this measure would increase the continuously available flow to the East Poplar River, the level of TDS discussed in option (iv) would not be improved except during spring runoff and following rainfall events. Moreover, the collector and main pipelines would have to be of considerably greater size in order to accommodate runoff flows.

The possibility of oil and grease contamination and suspended sediment in mine sump water suggest that this effluent should be contained for settlement before mixing with other waters or discharging into the East Poplar River. In addition, the size of the runoff retaining sums in intercepted drainages could be increased, permitting more modest sizing of diversion pipes and lower pumping rates for the diversion of high-quality surface runoff water. This remains a viable possibility, but would offer only marginal increases in overall water quality during the problem season from May to September.

- (vi) Treatment to reduce total dissolved solids and boron and diversion of all mine effluents to the East Poplar River downstream from Morrison Dam.

By incorporating reverse-osmosis water treatment in conjunction with collection and diversion measures discussed in option (v), a significant and continuous flow of very high quality water in the East Poplar River could be achieved. This volume could be as large as 12 percent of the reservoir volume; however, as with other proposed water treatment options, the cost to install and operate such a facility would be very high. The improvement in water quality that could be achieved, though significant, does not appear to warrant the extensive investment required to install and operate such a facility.

- (vii) Collection of mine effluents and diversion with controlled outfall.

An option which would allow selective diversion of mine effluent waters into Cookson Reservoir or into the East Poplar River downstream from the reservoir has been considered. This measure would incorporate a weir and control structure to selectively divert the mine effluents to either outfall depending on desired flows and water quality. Although this would guarantee flows in the Poplar River, the expected TDS content of effluents would offer only a marginal improvement, if any, in water quality.

- (viii) Increase in rate of mine dewatering.

A proposed increase in the rate of mine dewatering would offer increased flows available to the reservoir and hence, to the East Poplar River. This option would be effective only in periods of low flow and when high levels of TDS are experienced in the reservoir. At other times, the level of TDS present in the dewatering discharge would contribute to deterioration in water quality; therefore, this does not appear to be a very useful possibility.

- (ix) Reclamation and runoff control procedures to improve water quality.

Measures included in this option would retain runoff from lands undergoing reclamation in order to limit sedimentation in Girard Creek and to limit the amount of nitrate from fertilizers that could enter the stream and the reservoir. Specific operational procedures to control sedimentation

are recommended in Sections 3.6.3 and 3.6.4. Once vegetation has been re-established on the land, the loss of sediment should be minimal.

#### 8.4.2 Ash Disposal System Alternatives

As determined by the sensitivity analyses conducted for the proposed operation in Section 8.2, control of pollutant loadings from the ash lagoons offer the greatest potential for improving water quality at the International Boundary. Of the ash disposal systems and control measures listed in Section 8.3.3, the following represent the combinations that were examined for effectiveness in reducing TDS and B loadings to water crossing the border.

- (i) Proposed combined ash, once-through lagoon system with boron removal from the decant.

A selective resin for boron removal called Amberlite is available from Rohm and Haas Company. Activated carbons, such as Filtrasorb 300 manufactured by Calgon Corporation, have also been found to achieve a boron removal efficiency of about 90 percent if the initial concentration of boron in the solution does not exceed 5 mg/l (Choi and Chen, 1979). If coupled with improved seepage control through use of better lagoon lining, pump back of seepage losses or limiting the distance southward the lagoons can extend, sufficiently low B levels at the border might be achieved with treatment. Boron removal would not reduce the added TDS from the lagoons so any TDS reduction at the border would have to be achieved through some other means. Also, a very large installation would be needed to treat the ash lagoon decant and large quantities of sulphuric acid ( $H_2SO_4$ ) and caustic (NaOH) would have to be used for resin regeneration. Wastes from the treatment facility would have to be contained and disposed of in an acceptable manner. Nevertheless, this option was examined for its potential with boron removal down to 1.0 mg/l in the decant assumed.

Boron removal costs using the Amberlite resin were estimated for ash lagoon decant flows of 140 l/s (4 cfs) and 280 l/s (10 cfs), which are the approximate 1 unit and 2 unit decant rates. Three 15 ft diameter resin columns 10 ft high would be required to treat the 140 l/s flow, with 6 columns required for 280 l/s. The vessels would be rubber lined steel and the estimated capital cost complete with the necessary accessories such as valves and pumps, resin and waste containment pond are:

140 l/s	\$1,825,000
280 l/s	\$3,650,000

The following approximate annual costs for 2 units with a once-through lagoon system and B removal from the decant would result:

<u>Ash Disposal System (\$/year)</u>	<u>Increase over Proposed Ash Disposal System (\$/year)</u>	<u>Increase in Project Annual Cost (Percent)</u>
With lagoon lining as proposed		
2,677,730	623,960	30.5
With improved lagoon lining (bentonite or clay)		
2,971,590	927,820	45.4 - 178.1
-3,639,320	-1,595,550	1.40 - 2.41

(ii) A combined ash, recirculating lagoon system

The Poplar River ash disposal system has been designed with the option of recirculating ash transport water from the lagoon and successively reusing the water for further ash transport. This option has been made available by the provision of level controlled ash sluice water system tanks which can be supplied with either reservoir water or returned lagoon water.

Such a system would have the advantage of requiring that only one ash lagoon cell be used at a time instead of using two or three cells in rotation as required with the proposed once-through system. It is probable that a recirculating system would be operated to maintain a minimum water depth above the ash level in the lagoon. This could be accomplished with a variable head release structure in a section of the lagoon remote from the slurry discharge pipe. Water might be continually discharged via this structure into a separate recovery and return area located so that it could effectively service a number of lagoon cells. From the recovery area, the water would be pumped back to the station for re-use in transporting ash.

The main problem with this system would likely be calcium sulphate build-up leading to scaling of pipes and other equipment. The system may require periodic blowdown to maintain acceptable concentration levels and make-up to replenish losses due to blowdown, evaporation and seepage. With the planned mode of operation at Poplar River, the releases from the main boilerhouse sump to the ash lagoon would more than satisfy probable make-up requirements. Much of the main boilerhouse sump release is ash system cooling water, and may be of a quality acceptable for make-up. The sump effluent should have an SO<sub>4</sub> level similar to reservoir water in most cases. Consideration might be given to extracting ash system cooling water prior to the main boiler house sump. If ash system cooling water

quality is such that it would be an undesirable make-up source, make-up water could be taken from the auxiliary or main cooling water discharges.

A combined ash recirculating system would have a number of advantages in addition to reducing direct releases of ash lagoon water to the reservoir. With the need to operate only one lagoon cell at a time, both evaporation losses and seepage losses are reduced. Evaporation could be reduced from the estimated average of 800,000 m<sup>3</sup>/year for three lagoons to 270,000 m<sup>3</sup>/year with only one lagoon. The smaller area contributing to seepage at any one time, the ability to maintain a lower average head on the lagoon by allowing the water level to rise as the ash level rises and not having to operate under maximum head before some ash and sediment has been deposited in the lagoon all contribute to a reduced seepage rate.

Although the combined ash recirculating lagoon may have less seepage loss, the quality of the seepage water is likely to be considerably poorer on average than that in the once-through lagoon. Through examination of maximum monthly elemental concentrations obtained in the ash lagoons in the computer runs described in Section 8.2.5 and observed concentration levels in existing recirculating ash systems at the Nanticoke generating station in Ontario (Landolt et al, 1977) and the Colstrip generating station in Montana (Radian Corp., 1977), the following concentration levels were assumed to be representative of levels that could occur in a combined ash recirculating lagoon at Poplar River:

<u>Element</u>	<u>Concentration (mg/l)</u>
Ca	600
Mg	500
Na/K	700
C1	70
T.Alk	800
SO <sub>4</sub>	1500
B	50

Numerous combinations of control measures for such a system are possible including operation with no blowdown (the proponents of the recently approved recirculating system for the Keephills station in Alberta (Energy Resources Conservation Board, Alberta, 1978) suggest that system will operate with no blowdown), limited blowdown, treatment of blowdown and various methods of seepage control.

The SPC has estimated that the additional cost involved in converting the presently proposed once-through system to a combined ash recirculating system would be approximately \$1,600,000. The annual costs of a recirculating lagoon system with different lining methods would be approximately as follows:

<u>Ash Disposal System (\$/year)</u>	<u>Increase over Proposed Ash Disposal System (\$/year)</u>	<u>Increase in Project Annual Cost (Percent)</u>
With lagoon lined as proposed		
2,131,490	87,720	4.3
With improved lagoon lining (bentonite or clay)		
2,313,220	269,450	13.2 - 31.6
-2,689,650	-645,880	0.41 - 0.97

(iii) Dry flyash disposal with a once-through bottom ash lagoon

With this system, flyash would be collected and transported dry to a suitable disposal area and the bottom ash would be slurry transported to a lagoon. Since the ash at Poplar River is expected to be made up of 23% bottom ash and 77% flyash, the water required for transport and the resulting decant is substantially reduced from the combined flyash/bottom ash slurry system. In addition, the problems with suspended solids and with trace contaminants which often concentrate in flyash, are considerably less. The dissolution rate of boron from bottom ash is normally found to be less than from flyash and the concentrations that might be expected in a bottom ash lagoon might, therefore, be expected to be less. In the ash leachate study done for the Boundary Dam station, B concentrations in the bottom ash leachate were found to be less than half and often as low as 20 percent of the levels in the flyash leachate. The lower decant rate would also make boron removal from the decant more feasible. Sulphate and TDS levels can, however, be expected to be as high and perhaps higher than those in a combined ash lagoon.

In addition to reducing both the TDS and B additions to the reservoir, seepage and evaporative losses would be reduced. Operation could probably be accomplished using 2 lagoon cells for alternate filling and settling with each cell less than half the size of those for the proposed combined ash system. This would reduce the evaporative loss to less than 1/3 the 2 unit rate and reduce the area contributing to seepage at any one time to less than 1/3 the combined system contributing area.

The costs for this ash disposal alternative would not be substantially different from those estimated in (v) below for the dry flyash, dewatered bottom ash system.

(iv) Dry flyash disposal with a recirculating bottom ash lagoon

This option would provide additional control over (iii) above by limiting direct discharge back to the reservoir to blowdown requirements and permitting operation with only one lagoon cell at a time and a water level that could be kept down until some ash and sediment has been deposited in the lagoon.

(v) Dry flyash and dewatered bottom ash mechanically transported to a remote landfill

This alternative requires no ash lagoons. It is assumed that the flyash is collected dry, bottom ash is collected wet and dewatered and the two types of ash are subsequently transported by truck, rail or conveyor to separate or combined landfills. Bottom ash is typically dewatered through use of hydrobin ash-water separators although some new Canadian stations may have bar-link or scraper conveyor bottom ash collectors with water used for submergence or cooling sprays only. The scraper conveyor systems offer considerable capital and operating cost saving over wet hoppers and dewatering systems.

The most significant drawbacks of this disposal method are locating a suitable landfill disposal site and operational and economic considerations. The landfill site would have to be located to ensure ash placement above the water table and seepage into a larger more generalized groundwater flow system. The landfill area would be prepared in advance, provided with a perimeter and ash surface drainage and collection system and reclaimed as ash placement proceeds.

This type of ash collection and transport is similar to that used now at some existing Canadian generating stations. Examples are Lakeview in Ontario and Sundance in Alberta which transport dry flyash and dewatered bottom ash by truck. A conveyor system has been proposed (Integ-Ebasco, 1977) as part of the "zero discharge" waste management system for the Hat Creek station in British Columbia. Water use is limited in the Hat Creek proposal to that required to ensure dust suppression, which is largely accomplished with the wet bottom ash. The designers state that "no unusual or unproven technology is required" even though they are considering ash quantities far greater than those at Poplar River.

A dry flyash/dewatered bottom ash system would minimize water lost in ash disposal. The Hat Creek report (Integ-Ebasco, 1977) suggests that water lost with the dewatered bottom ash would be about  $0.26 \text{ m}^3/\text{tonne}$  (500 lb/ton) and Saskmont Engineering Ltd. suggests that total water lost with such a system would be about 2.3 l/s. These estimates convert to a water loss at Poplar of  $2520 \text{ m}^3/\text{month}$  and  $6070 \text{ m}^3/\text{month}$  respectively for 2 units. These losses are only 4 and 9 percent of the predicted evaporative loss of  $800,000 \text{ m}^3/\text{year}$  for the proposed ash disposal system.

An approximate capital cost for a dry flyash, dewatered bottom ash system is estimated to be \$16,350,000. This converts to the following annual costs.

<u>Ash Disposal System (\$/year)</u>	<u>Increase over Proposed Ash Disposal System (\$/year)</u>	<u>Increase in Project Annual Cost (Percent)</u>
3,171,830	1,128,060	55.2

Estimates for the Keephills power plant (Energy Resources Conservation Board, Alberta, 1978) suggest the cost of a similar system there will be about \$6.50 per tonne of ash. With disposal of 474,320 tonnes/year of ash, at Poplar River, the annual costs at \$6.50 per tonne are \$3,083,080/year, which is approximately the same as the cost given above.

#### 8.4.3 Control of Seepage from Ash Lagoons

There are numerous means available to control seepage from the ash lagoons at Poplar River. The method being proposed by the SPC consists of digging up the upper 0.6 m of till and recompacting to achieve a maximum permeability of  $1.5 \times 10^{-5}$  cm/s. More effective seepage control might be achieved by reworking a thicker layer or by hauling in different base material such as bentonite or clay. A layer of bentonite worked into the surface layer of compacted till would likely lower permeability of the lagoon base to the  $10^{-6}$  cm/s to  $10^{-7}$  cm/s range. Bentonite suitable for lining is presently being mined by Avonlea Mineral Industries Ltd. at Truax, Saskatchewan, approximately 90 km north of the Poplar River plant site. The estimated costs for both the proposed method of lining and improved lining are included above in the cost estimates for the lagoon ash disposal alternatives.

Another lining alternative would be the use of one of the many plastic membrane liners available on the market. A plastic lining if installed over the full 7 lagoons ( $3,793,000 \text{ m}^2$ ), is likely to cost between \$15,000,000 and \$20,000,000 (subdrain costs not included), or four to six percent of the two unit plant cost. The largest known plastic liner application in the world, the lining and subdrain system for a  $650,000 \text{ m}^2$  uranium-vanadium tailings impoundment in Colorado, is estimated to cost \$6,080,000. The use of a plastic membrane liner for the full ash lagoon area at Poplar River is unlikely.

A drainage system consisting of a gravel layer beneath the liner would ensure that seepage would be prevented from moving downward to the Empress Group. The gravel layer would be placed over a compacted till base graded so that seepage water would drain to a network of collection ditches or tiles. The collected seepage would be pumped back to the lagoon. The cost of a clay or bentonite liner with under drainage is estimated at \$5.00/m<sup>2</sup> to \$10.00/m<sup>2</sup>. At \$10.00/m<sup>2</sup> the cost for one 500,000 m<sup>2</sup> recirculating lagoon is \$5,000,000 and for the full 7 lagoons ( $7,793,000 \text{ m}^2$ ), \$37,930,000. Although the cost of installing a full liner/drainage system on an area as large as this may be impractical, if a lagoon disposal method is adopted it would be desirable to install at least a test section of the lagoon area with a drainage system as a means of

demonstrating the effectiveness of the selected lining and determining whether more stringent control, such as dry flyash disposal, is needed.

Locating the ash lagoons such that direct seepage flow to the East Poplar River below Morrison Dam does not occur would help minimize the amount of deterioration of border water quality as well as reduce concern over possible errors in seepage flow estimates. With either the reworked upper till or clay lining options, a good deal of uncertainty regarding seepage rates is likely to remain because of the fractured nature of the till beneath the lining, the probability that the lining material will also become fractured if not kept saturated during use and after retirement of the lagoon, and the uncertainties in estimates of the Empress Group carrying capacity. If the lagoons were located, for instance, in the Y-shaped piece of land between Girard Creek and the East Poplar River, all seepage flows would be to the reservoir and many of the uncertainties would not exist. Any location north of the power plant would be more suitable from the point of view of minimizing the potential for seepage to the East Poplar River.

If the proposed ash disposal area southeast of the station is used, a measure of seepage control might be achieved by (i) providing a horizontal separation between the ash lagoon perimeter dikes and the cooling water discharge channel and (ii) limiting the southern extension of the ash lagoons to keep the disposal area away from the higher hydraulic gradients to the East Poplar River in the Empress Group formation. A separation of 50 m between the toe of the ash lagoon dikes and the access road along the west side of the cooling water channel with a seepage collection ditch along the separating strip of land would be desirable. A southern limit to ash lagoon construction at the 1000 m North location on the project site grid (see Figure 5-1) would put the southern boundary of the lagoons about 700 m north of Morrison Dam and beyond the zone of steepest hydraulic gradient.

Another factor that should be mentioned in consideration of seepage control requirements is attenuation of B in the soils of the upper till layer beneath the lagoons. Maximum adsorption of B by soils is suggested (Choi and Chen, 1978) to be about 2 mg of B/g of soil. Recent adsorption tests conducted by the SPC on Poplar River ash lagoon area soils indicated a B adsorption capacity of 11 ug/g of soil. The time required to use up the B adsorption capacity of the upper till layer (15 m) with 50 mg/l B in the seepage flow and different seepage rates are listed below:

<u>Seepage Rate (l/s)</u>	<u>Time to Deplete B Adsorption Capacity (Years)</u>
2	42
5	17
10	8
20	4

Since each recirculating lagoon cell will be subjected to an operating hydraulic head for 12 years with 1 unit and 6 years with 2 units, the B adsorption capability of the till may be more than adequate to attenuate the B during the active lagoon life if the seepage is controlled.

The retention of B by soils is thought to be closely related to the presence of hydroxy forms of Fe, Al and Mn (Choi and Chen, 1979, Sims and Bingham, 1968). Retention varies with pH and is usually maximum in the 6 to 9 pH range, the optimum pH being different for different hydroxy metal compounds. Above a pH of about 9, the B retention rate is markedly reduced. The Groundwater Quantity and Quality Committee has suggested that the pH of ash lagoon seepage waters is likely to reduce to approximately 8 during downward leaching into the till due to the precipitation of metal hydroxides and hydroxyl exchange capacity of the till. Since a pH of 8 is in the optimum range for B retention, the adsorption rate of 11 ug/g of soil, measured at a pH of 9 in the soil adsorption tests, is probably a reasonable estimate for retention in actual operation. However, because of the high dependence of B retention on optimum pH, no retention has been assumed in the reservoir operation model computer runs. Some retention can be expected in actual operation and this modelling assumption of no retention is a very conservative or worst case assumption. Also, the use of 50 mg/l B in the seepage water could well be higher than the concentration that will actually be experienced and probably adds another element of conservatism to the computer output results.

In the computer analyses, seepage flows are assumed to reach the East Poplar River in same month that they occur. In actual practice they are likely to take at least 10 years and probably much longer to travel the distance from the ash lagoons to the river. They will also spread horizontally to some degree in the Empress Group sand and gravel layer. This will provide some opportunity for dilution with groundwater from the west and tend to even out any peaks or lows. Hence, the assumption of direct and immediate discharge is again a worst case assumption. A displacement in time, however, will mean that seepage flows will in reality continue beyond the life of the project.

#### 8.4.4 Effectiveness of Ash System Mitigation Alternatives

A number of runs of the reservoir operation model were made to determine the effectiveness of different ash disposal system alternatives. Since the mine dewatering could have a more detrimental affect on water quality with some of the ash lagoon mitigation alternatives than it was found to have with the proposed system, several computer runs were also made to determine the effects of a higher dewatering rate. Table 8-3 lists the water quality levels computed for the International Boundary for each of the alternatives examined and commentary on each alternative is provided below.

TABLE 8-3 RESULTS OF RESERVOIR OPERATION MODEL RUNS FOR PRIMARY MITIGATION OPTIONS

NO.	COMPUTER MODEL RUN DESCRIPTION	WATER QUALITY AT INTERNATIONAL BOUNDARY						IDENTIFICATION OF CONCENTRATION LISTED	
		Na/K (mg/l)	Ca (mg/l)	Mg (mg/l)	T. Atk. (mg/l)	SO <sub>4</sub> (mg/l)	Cl (mg/l)	TDS (mg/l)	B (mg/l)
12	2 units; avg. leachate rate; avg. seepage (3.5 l/s per ash lagoon); boron treated to 1/mg/l in ash lagoon decant	518 (183) 388 189	388 (176) 298 178	452 205 367 212	795 424 685 435	1162 357 817 370	46.9 13.3 32.6 13.9	2468 927 1857 957	9.2 (2.2) (2.5) (1.8)
11	2 units; no ash lagoons; 2500 m <sup>3</sup> /month water loss with bottom ash	445 177 351 191	388 190 305 191	457 221 380 229	1063 519 878 535	698 266 538 275	32.4 11.0 24.0 11.3	2079 895 1655 923	3.9 1.7 3.1 1.7
14	2 units; no ash lagoons; 2500 m <sup>3</sup> /month water loss with bottom ash; mine dewatering continuous at 100 l/s	365 (200) (340) (200)	344 (220) (320) (220)	376 (230) (360) (230)	926 (600) (875) (575)	541 (280) (510) (290)	20.8 (10.0) (19.0) (10.5)	1707 (975) (1600) (975)	3.1 (1.8) (2.9) (1.8)
13	2 units; combined ash recirc. lagoon; no blowdown; seepage of 3.5 l/s to river, 0.0 l/s to reservoir; specified seepage concentrations	470 (200) (398) (190)	410 (210) (344) (210)	469 (230) (416) (230)	1060 (550) (945) (550)	769 (320) (631) (300)	35.8 (13.0) (29.4) (13.0)	2189 (950) (1864) (900)	7.2 (3.5) (4.8) (2.8)
17b	2 units; combined ash recirc. lagoon; no blowdown; seepage of 1.5 l/s to river; 5.0 l/s to reservoir; specified seepage concentrations	481 187 374 193	420 197 325 198	476 225 392 232	1071 518 884 533	794 293 598 300	36.8 12.2 26.8 12.4	2237 936 1754 961	7.9 2.9 5.6 2.9
20	2 units; combined ash recirc. lagoons; blowdown of 15 l/s; seepage of 1.5 l/s to river, 5.0 l/s to reservoir; specified seepage concentrations	546 210 422 216	476 215 368 217	509 237 529 244	1085 349 901 544	965 14.7 32.3 15.0	44.3 1035 1967 1060	15.3 4.2 6.5 4.3	Max. Month Mean Annual Max. May to Sept. Mean May to Sept.

NOTE: All bracketed figures are approximated from earlier output listings which did not provide data in the exact form required.  
All concentrations listed except the maximum monthly values are flow weighted.

(a)      Dry Flyash/Dewatered Bottom Ash

It can be seen from Table 8-3 that boron levels at the International Boundary would remain well under the recommended limits with a dry flyash/dewatered bottom ash system (Runs Nos. 11 and 14). However, the maximum May to September mean TDS of 1655 mg/l is above the irrigation season limit for any consecutive 90-day period of 1500 mg/l. An examination of the complete output listing for Run Nos. 11 and 14 shows that the 1500 mg/l limit is exceeded for 4 periods of 3 months in the 46 years simulated, these being in 1936, 1937 and 1938 at the end of the extreme 1930's dry period for both runs. The addition of more water from mine dewatering (Run No. 14 did not significantly change the border TDS levels. Also noted from the Run No. 11 output, the May to September 10-year running mean exceeded the 1000 mg/l limit for 11 10-year periods in the 46-year simulation period, with a maximum 10-year mean of 1194 mg/l.

(b)      Removal of Boron from Decant

The recommended B objectives for the border can also be met by treating the decant to remove B from the proposed once-through ash lagoon system decant (Run No. 12). However, this does not improve the border TDS levels. The 3-month mean exceeds the 1500 mg/l limit for 7 3-month periods during 1936-38 and the 10-year running mean TDS exceeds 1000 mg/l for 12 10-year periods in 46 years. The frequency with which the 1500 mg/l single season limit is exceeded is slightly greater than for the dry flyash/dewatered bottom ash system and the higher average level increases by one the number of long-term (10-year) mean values above 1000 mg/l.

(c)      Combined ash Recirculating Lagoon

Run Nos. 13 and 17b simulate a combined flyash/bottom ash recirculating lagoon system with no blowdown or blowdown treated to at least the existing reservoir water quality level. Run No. 13 assumes seepage from the ash lagoons is directed to the East Poplar River only at a rate of 3.5 l/s whereas Run No. 17b simulates seepage rates of 5 l/s and 1.5 l/s to the reservoir and the East Poplar River respectively. The recommended border levels for B can be achieved with the combined ash recirculating system with either of the seepage assumptions. The TDS levels, as expected, are slightly higher than for the dry flyash/dewatered bottom ash alternative because of the increased evaporative loss from the lagoon and TDS contributions from the seepage flows. Since the Run No. 13 output listings did not contain all pertinent statistics, only the Run No. 17b output was examined in detail. That information shows that the 3-month irrigation season limit for TDS of 1500 mg/l was exceeded during 4 3-month periods in the 46 years simulated. These were during the 1936-38 period.

The 10-year irrigation season limit of 1000 mg/l, however, was exceeded during 12 of 46 10-year periods with a maximum 10-year mean of 1217 mg/l. Detailed output listings from Run No. 17b for TDS and B are provided in Annex J.

If blowdown from a combined ash recirculating lagoon was not treated, the total ash lagoon water returned to the reservoir at lagoon water quality would be higher than the 5 l/s average seepage flow return. The results of Run No. 20 simulate the resulting water quality from a 20 l/s untreated discharge (assumed to be a combination of seepage and blowdown) from the ash lagoon to the reservoir. This mode of operation would have a substantial detrimental effect on both the B and TDS levels. The mean B level increased to 5.2 mg/l from the 2.9 mg/l in Run No. 17b and the 5 mg/l 10-year limit was exceeded 25 times. The 2-consecutive month limit of 8 mg/l was also observed to be exceeded during several years in the simulated 1930's period. A similar situation occurred with TDS with the number of exceedances of the 10-year limit increasing from 12 to 27 and of the 3-consecutive month limit from 4 to 9. Hence, it is important not to have untreated blowdown releases from the ash lagoon to the reservoir in attempting to meet the recommended border limits.

Several additional computer runs, for which results are not presented in Table 8-3, were made to determine the sensitivity of border water quality to seepage rate to the East Poplar River. The sensitivity runs showed that with the combined ash recirculating system, seepage flows greater than 2 l/s give maximum monthly B concentrations above 8 mg/l at the border and a mean May to September B greater than 3 mg/l suggesting that the seepage rate should not be permitted to exceed about 2 l/s with that method of disposal.

#### 8.4.5 Supplemental Mitigation Options for TDS Reduction

##### (a) Evaluation of TDS Objectives and Loadings

It is apparent in examining the results of the above computer model sensitivity runs for different mitigation options that although recommended border levels for B can be met, it would be difficult to achieve either the single season or the 10-year TDS levels with any of the options considered if an extended dry period such as the recorded 1930's period were to reoccur. One might first question the recommended border levels since under natural conditions essentially all flows crossing the border during the May to September period for a number of years in succession during the 1930's would have been from groundwater supply having a TDS level of 1100 mg/l or higher. The maximum recorded TDS in the East Poplar River at the International Boundary during summer was 2250 mg/l and the average summer value is listed (Surface Water Quality Committee, Water Quality Data, Seasonal and Monthly Summary, May, 1978) as 1018 mg/l.

Applying the TDS versus flow regression equation developed by the Surface Water Quality Committee to the present use flows (Table 6-2) at the International Boundary during the 1930's period, with an upper limit of 2250 mg/l, shows expected TDS levels during that period under natural conditions to be:

<u>Year</u>	<u>May</u>	<u>June</u>	<u>July</u>	<u>Aug.</u>	<u>Sept.</u>	<u>Mean</u>
1931	858	1678	1678	1967	838	1404
32	755	1039	1207	278	838	823
33	454	438	626	914	1138	714
34	929	2190	2250	2250	1326	1789
35	734	510	663	1277	1577	952
36	595	1138	1577	1520	1381	1242
37	599	1016	755	1577	1277	1045
38	567	747	702	1445	1055	903
39	696	406	1326	2250	1381	1212
1940	573	702	796	521	885	695

It can be seen from the table that 5 of the 10 May to September mean TDS levels from 1931-40 exceed 1000 mg/l and several 3-consecutive month periods have mean TDS levels above 1500 mg/l. The 10-year mean for this period is 1078 mg/l and the 10-year means encompassing a number of years before and after this period would probably have had 10-year irrigation season means greater than 1000 mg/l. Each of the years in the 1931-40 period did, however, have flows averaging about 1.54 m<sup>3</sup>/s during the maximum spring inflow month, which according to the regression equation would have produced an inflow TDS of about 260 mg/l. Water of this quality might have been stored downstream during the spring for subsequent irrigation use if the Poplar River project had not been undertaken. This presumably is the basis for the recommended 10-year TDS limit of 1000 mg/l.

In most years, the peak runoff period in the spring results in high flows over several days rather than the entire month, with those days contributing a large portion of the annual inflow to the reservoir. The TDS vs monthly flow ( $Q_m$ ) regression equation developed by the Surface Water Quality Committee is an adjustment to a TDS vs instantaneous flow ( $Q_i$ ) equation determined from recorded data for 1973 and 1974. To determine whether the TDS vs monthly flow accurately accounts for the variable peak inflow conditions, total TDS loadings were computed separately with both the monthly and instantaneous flow equations and compared. The equations used are:

$$\text{Monthly flow} \quad \text{TDS (mg/l)} = 1418 Q_m^{-0.45} + 20$$

$$\text{Instantaneous flow} \quad \text{TDS (mg/l)} = 1322 Q_i^{-0.389} + 20$$

The total TDS loading for the 46-year period determined by applying the monthly equation was  $203.4 \times 10^6$  kg. The total computed by applying the instantaneous flow equation to daily flows during the high month each year and the monthly equation to all other months was  $202.2 \times 10^6$  kg, only 0.6 percent lower than the quantity obtained by applying the monthly equation to all months. Hence, on average the monthly equation would appear to provide a good estimate of the TDS loading to the reservoir from natural inflows.

(b) Supplemental Mitigation Measures

Several supplementary measures could be undertaken in conjunction with either the dry flyash/dewatered bottom ash, the combined ash recirculating lagoon or the dry flyash/recirculating bottom ash lagoon alternatives to bring the border TDS levels down to the recommended limits. Several computer model runs were made with the addition of specific supplementary options to the combined ash recirculating system alternative. The results are contained in Table 8-4. The supplementary options examined are:

- (i) lime soften the reservoir water;
- (ii) decrease forced evaporative loss from the reservoir by increasing the period of planned power plant outage;
- (iii) lower the TDS of the mine dewater by softening or some other means;
- (iv) supply the release-on-demand flow to the East Poplar River with better quality water stored and diverted annually from the Poplar River;
- (v) store the East Poplar River spring runoff flows from freshet to the release-on-demand month by means of a reservoir upstream of Cookson Reservoir and use this water later to supply all or part of the release-on-demand.

Lime Softening

Lime softening the reservoir water (Run No. 21b) as a supplement to the combined ash recirculating lagoon alternative was found to be effective in reducing TDS levels at the border. With the level of softening needed to prevent condenser tube scaling, the mean annual TDS level at the border is reduced from 920 mg/l to 828 mg/l and one 3-month mean exceeds 1500 mg/l. The 10-year running mean TDS is greater than the 1000 mg/l limit for 7 of the 46 10-year periods with the highest 10-year mean being 1054 mg/l. Detailed results for TDS and B are provided in Annex J.

TABLE 8-4 RESULTS OF RESERVOIR OPERATION MODEL RUNS FOR SUPPLEMENTAL MITIGATION OPTIONS

NO.	COMPUTER MODEL RUN DESCRIPTION	WATER QUALITY AT INTERNATIONAL BOUNDARY						IDENTIFICATION OF CONCENTRATION OF LISTED		
		Na/K (mg/l)	Ca (mg/l)	Mg (mg/l)	T. Alk. (mg/l)	S0 <sup>4</sup> (mg/l)	C1 (mg/l)	TDS (mg/l)	SAR	B (mg/l)
21b	2 units; combined ash recirc. lagoon; seepage of 1.5 l/s to river, 5.0 l/s to reservoir; reservoir water lime softened	448 354 190 255 184	185 155 132 141 131	334 301 208 240 201	608 559 438 483 423	693 539 285 385 277	34.0 25.2 12.2 17.4 12.0	1660 1375 853 1054 828	9.1 7.5 4.4 5.8 4.3	7.5 5.4 2.9 3.8 2.9
22	2 units; combined ash recirc. lagoon; seepage of 1.5 l/s to river, 5.0 l/s to reservoir; mine dewater conc. reduced by 50%	407 324 175 236 170	344 274 214 212 177	404 344 488 587 474	902 767 272 369 265	667 515 11.9 17.0 11.7	33.1 24.5 11.9 17.0 11.7	1883 1517 876 1120 852	6.5 5.6 3.8 4.8 3.7	7.2 5.2 2.8 3.7 2.8
23	2 units; combined ash recirc. lagoon; seepage of 1.5 l/s to river, 5.0 l/s to reservoir; 1 unit down during June, July & Aug.	414 332 186 246 180	362 289 190 226 189	409 350 223 270 216	934 796 514 618 500	672 524 287 385 279	31.2 23.3 11.8 16.7 11.7	1921 1558 923 1174 898	6.5 5.7 3.9 4.9 3.8	7.0 5.1 2.8 3.7 2.8
24	2 units; combined ash recirc. lagoon; seepage of 1.5 l/s to river, 5.0 l/s to reservoir; demand release from Poplar River	431 253 163 195 172	375 225 160 181 180	426 288 209 236 213	971 667 495 554 496	700 390 243 297 266	32.2 15.5 8.3 11.0 10.5	1999 1218 825 958 869	6.6 4.8 3.6 4.1 3.7	7.2 3.1 2.1 2.6 2.6
25	2 units; combined ash recirc. lagoon; seepage of 1.5 l/s to river, 5.0 l/s to reservoir; storage above Cookson Reservoir	483 273 145 187 171	418 245 152 177 181	473 303 183 216 208	1063 692 426 502 480	799 423 219 283 265	37.2 17.1 8.1 11.1 10.7	2240 1300 733 904 857	8.0 5.0 3.3 4.0 3.7	4.1 2.2 2.8 2.7 2.7

The results indicate that with the lime softening supplementary options, recommended border TDS levels are very nearly met for the entire 46-year period. A further reduction would likely be achieved with softening beyond that needed to control scaling. Hence, the combination of a combined flyash/bottom ash recirculating lagoon and reservoir water softening is considered adequate to ensure that both the recommended B and TDS levels are met. Lime softening coupled with any of the dry flyash disposal alternatives would, therefore, also provide adequate protection. The method, however, has the disadvantage of posing an additional waste disposal problem and a retention pond would have to be provided.

The estimated annual costs for a combined ash recirculating lagoon system coupled with lime softening, including costs for sludge disposal from the softening facility are as follows:

<u>Ash System and Softening (\$/year)</u>	<u>Increase over Proposed Ash Disposal System (\$/year)</u>	<u>Increase in Project Annual Cost (Percent)</u>
With lagoon lining as proposed		
2,448,870	405,100	19.8
With improved lagoon lining (bentonite or clay)		
2,630,600 -3,007,030	586,830 -936,260	28.7 - 47.1 0.89 - 1.45

#### Extension of Planned Power Plant Outage

With this option the planned maintenance outage was assumed to be increased from 1 month to 1 1/2 months per unit. This in effect reduces the forced evaporation for the month of July from the 2-unit rate to the 1-unit rate of 203,100 m<sup>3</sup>. This brought about only a small drop in TDS levels as observed from the output from Run No. 23. The mean annual TDS level was reduced from 920 mg/l to 898 mg/l with 2 3-month periods during May to September exceeding the limit of 1500 mg/l as opposed to 4 excess periods with Run No. 17b. The 10-year irrigation season limit of 1000 mg/l was exceeded 11 times as opposed to 12 times with the regular planned maintenance outage. Hence, this option has some effect but shutdown of both units during the highest evaporation summer months would be needed before the TDS would be markedly reduced. Since water supply for hydro-electric power would be low during an extended dry period, it is unlikely that extended shutdown would be a viable option.

No cost estimate was made for this option.

#### Reduction in TDS of Mine Dewater

In Run No. 22, the concentrations of all elements in the mine dewater discharge to the reservoir was assumed to be reduced by 50 percent. The ease of achieving this has not been assessed in detail, but

presumably the dewater flow would represent an easily controllable and fairly constant quantity of water that could be lime softened or treated in some other manner. Treating the low estimate of mine dewater flow is more effective in lowering average TDS levels than the extended forced outage option examined. The mean annual TDS reduced from 920 mg/l to 852 mg/l, 2 3-month periods during May to September exceeded the 1500 mg/l limit and the 10-year limit of 1000 mg/l was exceeded 9 times with a maximum 10-year mean of 1120 mg/l. If the TDS could be further reduced in the mine dewater, this option would have some potential. The mean annual TDS of 852 mg/l is only 24 mg/l higher than the level achieved with the reservoir lime softening option. No cost estimate was made for this option.

#### Supply Release-on-Demand from Poplar River

This supplementary option assumes that sufficient spring freshet water would be stored on the Poplar River to meet the release-on-demand requirement on the East Poplar and, if desired, the riparian releases during the irrigation season. This water would be pumped across the divide to the East Poplar and released at the border when required.

Run No. 24 computes border water quality assuming that the release-on-demand flow each year is met by diverting Poplar River water with the following concentrations (mg/l):

Na/K	100	SO <sub>4</sub>	150
Ca (as Ca)	40	C1	4
Mg (as Mg)	40	B	.5
T. Alk (as CaCO <sub>3</sub> )	400	NO <sub>3</sub>	.2
		P	.02

These concentrations constitute an average TDS of about 575 mg/l which is probably much higher than average spring freshet TDS levels.

Examination of the output from Run No. 17b shows that the release-on-demand (R.O.D.) flows contribute the following percents of the total water volumes released across the border in the May to September period.

Year	Release-on-demand Volume (m <sup>3</sup> )	May to September Total Volume (m <sup>3</sup> )	R.O.D. as % of Total
1931	370,000	1,137,000	32.5
32	617,000	1,681,390	36.7
33	617,000	1,940,350	31.8
34	617,000	1,392,550	44.3
35	370,000	1,501,430	24.6
36	617,000	1,776,340	34.7
37	370,000	1,387,910	26.7
38	617,000	1,877,470	32.9
39	1,233,000	2,431,930	50.7
1940	617,000	1,943,730	31.7
Mean	604,500	1,707,010	34.7

The average 35 percent contribution of release-on-demand flow to total May to September flow suggests that a good quality release-on-demand should have a significant effect on the flow weighted May to September mean TDS at the border.

Run No. 24 shows that with a 575 mg/l TDS release-on-demand, and water that would otherwise have been used to meet the release-on-demand contained in Cookson Reservoir, the border TDS levels meet the recommended limits. The maximum single season May to September mean is 1218 mg/l occurring in 1937. All 10-year irrigation season means are well below the 1000 mg/l recommended limit with the maximum value of 965 mg/l occurring in 1933. Hence, it can be concluded that if water of the quality assumed could be retained each year during spring runoff on the Poplar River and diverted to supply the East Poplar River release-on-demand requirement, the quality limits for water crossing the International Boundary would be achieved. Also more favourable reservoir operating levels can be maintained with a minimum level during the 46-year simulation period of 747.98 m. Detailed TDS, B and reservoir level results for Run No. 24 are contained in Annex J.

The annual costs of containment and diversion facilities on the Poplar River, coupled with the annual costs of a combined ash recirculating lagoon system, are as follows:

Ash System and Diversion <u>(\$/year)</u>	Increase over Proposed Ash Disposal System <u>(\$/year)</u>	Increase in Project Annual Cost <u>(Percent)</u>
With lagoon lining as proposed		
2,238,440	194,670	9.5
With improved lagoon lining (bentonite or clay)		
2,420,170	376,400	18.4 - 36.8
-2,796,600	-752,830	0.57 - 1.14

#### Storage Upstream of Cookson Reservoir

With this supplementary option, all flow up to the design capacity of a small reservoir located upstream of Cookson Reservoir would be retained during the period from the spring freshet to July, the assumed month for the release-on-demand request. All or part of the release-on-demand-volume, depending on the inflow quantity stored, would then be conveyed from the upper reservoir via pipe to the riparian outlet at Morrison Dam and released downstream. Spring freshet flow in the East Poplar during low runoff years of the 1930's averaged about 1.54 m<sup>3</sup>/s which according to the TDS versus monthly flow equation would have a TDS level of about 260 mg/l. This level would be increased somewhat by storage of additional poorer quality

water between the time of the spring freshet and the time of the release-on-demand. It would also mean that the quality of water in Cookson Reservoir would be somewhat lower because of the reduced dilution by spring inflows.

Run No. 25 shows that if this supplementary option is coupled with a combined ash recirculating lagoon system or other ash disposal alternative with similar water pollution control capability, the recommended TDS limits at the International Boundary can be met. All 3-month periods during the irrigation season have mean TDS concentrations less than 1500 mg/l. The maximum 10-year mean for the May to September irrigation season is 904 mg/l, which is lower than the 1000 mg/l limit and also lower than the maximum value with diversion from the Poplar River. The improvement over the results obtained with diversion from the Poplar River is likely due to the fixed concentrations assumed for water diverted from the Poplar being higher than those computed from inflow rates for the retention on the East Poplar River. Detailed TDS, B and reservoir level results for Run No. 25 are provided in Annex J.

The results of Run No. 25 also show that a storage facility upstream of Cookson Reservoir, constructed and operated as assumed for the run, would not detrimentally affect reservoir operation. The minimum monthly computed water level in Cookson Reservoir is 747.23 m, essentially the same as the level of 747.20 determined for the combined ash recirculating lagoon system with no supplemental mitigation (Run No. 21b). It was assumed in the analysis that a rockfill weir would be constructed upstream of Cookson Reservoir with only enough retention capability to store the maximum release-on-demand required in the recommended apportionment (IJC, 1978) of 1,230,000 m<sup>3</sup>. All East Poplar River flows in excess of this amount would flow over the weir into Cookson Reservoir. Water would be retained in the upper reservoir until the release-on-demand was requested. After supplying the year's release-on-demand via gravity diversion pipe, any water remaining in the upper reservoir would be released into Cookson Reservoir via a small release works through the weir. This would minimize evaporative loss from the upper reservoir. Cookson Reservoir levels are not affected by this operation and could, in fact, be slightly improved because of the potential for retention in high runoff years of some of the water that would otherwise be spilled at Morrison Dam.

The estimated annual costs of a retention and diversion facility on the East Poplar River coupled with a combined ash recirculating lagoon system are as follows.

Ash System and Diversion <u>(\$/year)</u>	Increase over Proposed Ash Disposal System <u>(\$/year)</u>	Increase over Proposed Ash Disposal System <u>(Percent)</u>	Increase in Project Annual Cost <u>(Percent)</u>
With lagoon lining as proposed			
2,198,330	154,560	7.6	0.23
With improved lagoon lining (bentonite or clay)			
2,380,060	336,290	16.5	0.51
-2,756,490	-712,720	-34.9	-1.07

8.4.6      Comparison of Minimum Reservoir Levels with Selected Mitigation Alternatives

The following table lists, for comparison purposes, the minimum monthly water level computed for Cookson Reservoir during the 46-year period of analysis for some of the operation and mitigation alternatives examined.

<u>Run No.</u>	<u>Description</u>	<u>Minimum Reservoir Level (m)</u>
9	No units	750.04
6	1 unit, proposed ash disposal system	748.54
10b	2 units, proposed ash disposal system	745.42
10c	2 units, proposed ash disposal system, increased mine dewatering rate (100 l/s)	748.55
16	2 units, proposed ash disposal system, forced evaporation increased by 20 percent	744.73
11	2 units, dry fly ash/dewatered bottom ash system	747.22
17b	2 units, combined ash recirculating lagoon	747.20
24	2 units, combined ash recirculating lagoon with diversion from Poplar River	747.98
25	2 units, combined ash recirculating lagoon with storage upstream on East Poplar River	747.23

Figures 8-2 and 8-3 contain plots of 10-year running mean TDS and B levels for selected operation and mitigation alternatives. The 10-year mean is plotted on the first year of the 10-year period for which it was computed. The results plotted are for Run Nos. 10b, 17b and 25 described above and for run number 21b, which was for 2 units with a combined ash recirculating lagoon system supplemented by lime softening of the reservoir.

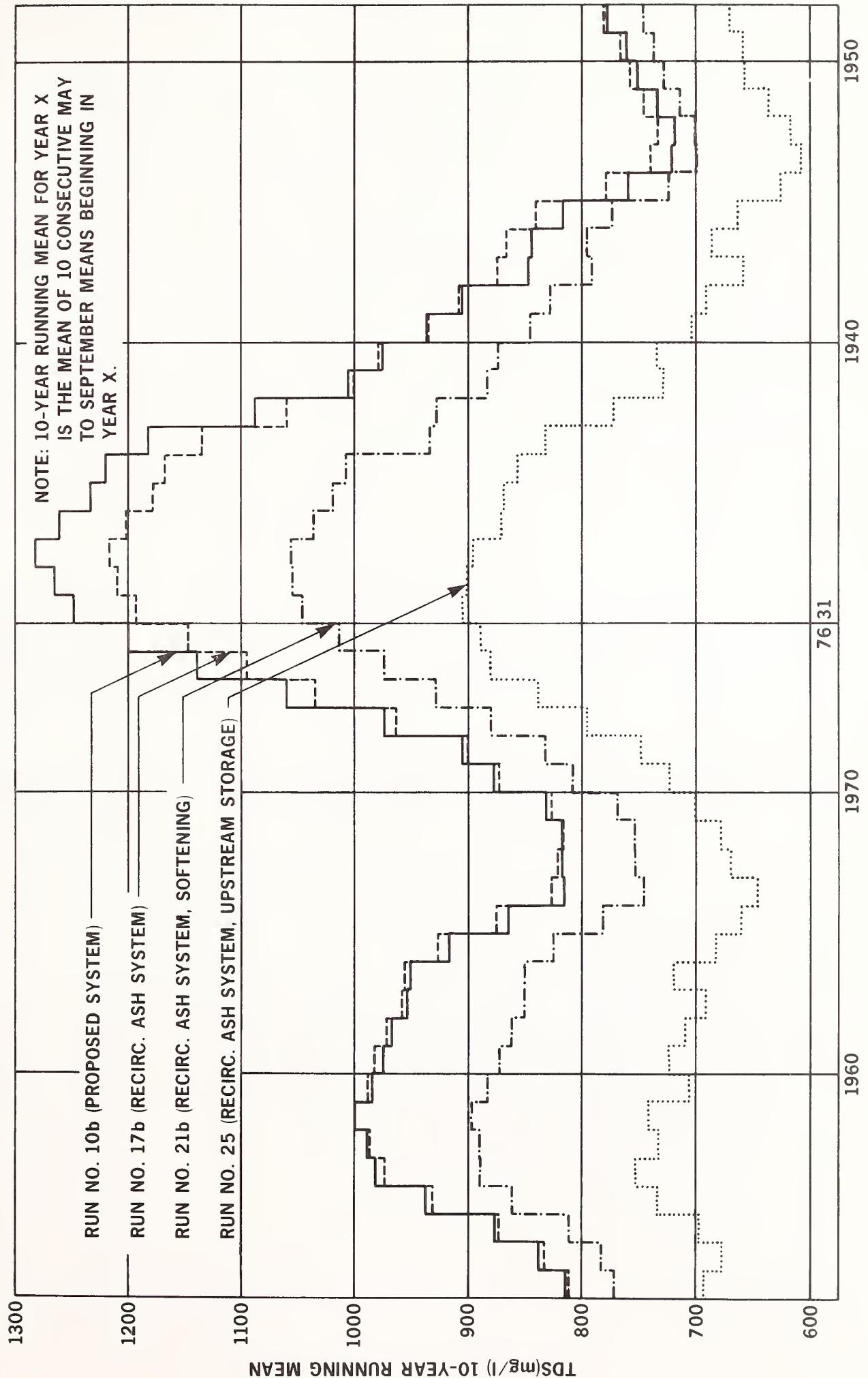


FIGURE 8-2 10-YEAR RUNNING MAY TO SEPTEMBER MEAN TDS LEVELS AT INTERNATIONAL BOUNDARY FOR SELECTED COMPUTER MODEL RUNS

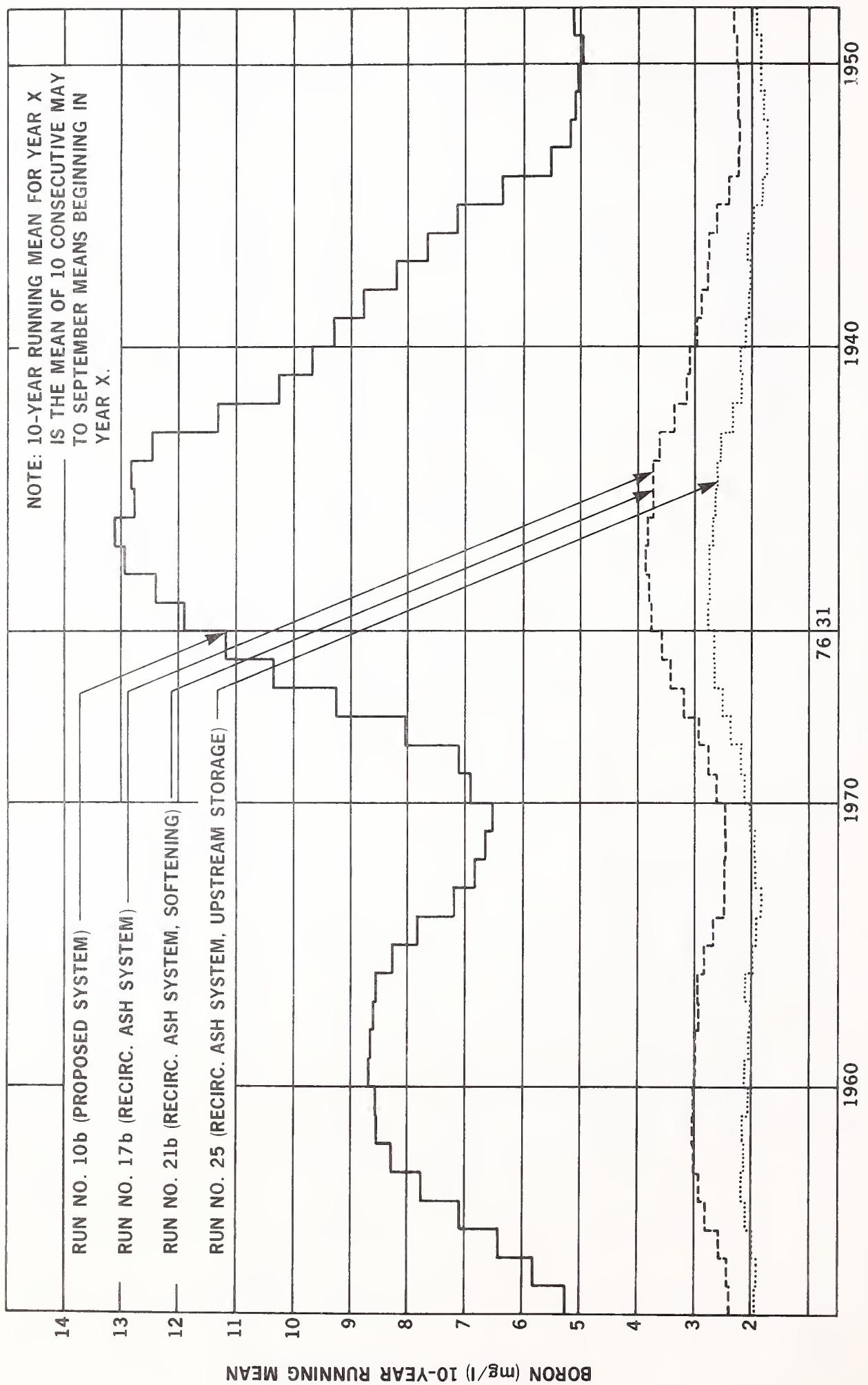


FIGURE 8-3 10-YEAR RUNNING MAY TO SEPTEMBER MEAN BORON LEVELS AT INTERNATIONAL BOUNDARY  
FOR SELECTED COMPUTER MODEL RUNS

#### 8.4.7 Mitigation to Meet Other Concerns

##### (a) Power Plant Wastewater Streams

If the combined ash recirculating lagoon or dry flyash/dewatered bottom ash systems were adopted, wastewater streams from the plant which were to have been discharged to the once-through ash lagoon would have to be disposed of by some other method. The entire wastewater management system for the plant would have to be re-examined. Sufficient treatment capability should be provided to ensure that release concentrations will not detrimentally effect reservoir water quality and subsequently border water quality. This might be achieved by use of equalization reservoirs or ponds in which the wastewaters, including ash lagoon blowdown if any is needed, could be retained before being treated. It is commonly recommended (Hykan et al, 1978) that wastestreams be segregated according to quality before treatment to enable efficient utilization of the treatment equipment with the need for metal removal one criterion for separation. Separate pre-treatment containment ponds could be provided for wastewaters needing metal reduction and those requiring primarily suspended solids reductions. No specific containment and treatment system is recommended but all effluents released should meet preset concentration limits designed to prevent reservoir water deterioration.

##### (b) Additions of Other Elements from the Ash Lagoons

Reference was made in Section 8.1 to possible concern over Cd, Cr, Mo and NO<sub>3</sub> as well as the B contributions to International Boundary levels of these elements. Concern over Cd is based on ash leachate test results whereas Cr, Mo and NO<sub>3</sub> have been identified only as typical elements of concern in ash pond waters.

In examining typical elements of concern in alkaline ash water, Se, B, Cr, Ni, Cu, Ba, As, Zn, and Al have been found (Chu and Ruane, 1978) to leach most readily. These are listed in order of the fraction of total trace metal content released from the ash. For neutral ash water the ranking by fraction of the element released was found to be B, Cd, As, Se, Zn, Ni, Mn, Cu and Ba. Elements found (Chu and Ruane, 1978 and Argonne, 1977) in excess of EPA water quality criteria were Ba, B, Cr, Ni and Se for alkaline ash waters and As, B, Cd, Mn, Ni and Se for neutral ash waters.

It is inappropriate to use typical leaching rates as more than a general guide on elements that might be of concern in a particular ash. The concentration of trace elements in ash is quite variable and conclusions should be based on actual analysis of dissolution rates with the ash being produced. The conditions in a recirculating ash lagoon could also lead to increased concentrations of certain elements due to the greater availability from the continuous new ash supply and a pH level dependent upon the relative buildup of constituents, particularly Ca and SO<sub>4</sub>, in

the lagoon. Of the constituents originally mentioned, Mo and  $\text{NO}_3$  are not often identified as concerns with ash lagoons.

The representative value in Table 5-7 for Mo leached from the Poplar River ash of 0.5 mg/l is, however, much higher than the 5 to 10 ug/l observed in groundwater samples. No border limit has been set for Mo. The  $\text{NO}_3$  increase of 0-2 mg/l in the ash lagoons is within the 10 mg/l recommended border objective.

The treatment methods used for removal of specific selected metals are also usually effective in reducing the levels of a variety of other trace elements. By treating to achieve prescribed effluent limits for several metals and periodic checking of the levels of other possible concern elements, adequate control could be exercised.

If direct inputs to the reservoir are controlled, seepage from the ash lagoon and other waste containment ponds may still be of concern. A study at the Michigan City G.S. showed that rapid attenuation occurred for most metals very close to the ash pond (Theis et al, 1978). Oxides of iron and manganese, once formed in the ash lagoon, became effective metal scavengers. However, curious peaks of metals were observed in the lagoon which could have lead to short-term higher concentration seepage discharge to surface waters. Also, metals were found to accumulate in the soil due to precipitation and adsorption onto the hydrous iron and manganese oxides.

To avoid accumulation in the soil, the extent and quantity of seepage should be controlled to the degree possible. The quantities of metals retained in solution in the Poplar River ash lagoons should be low because of the expected high pH. Care can be taken to ensure that pH remains fairly high at all times in a recirculating ash lagoon by controlled lime or limestone addition if necessary. This should not be necessary at Poplar River. However, even with good conditions for metal precipitation, accumulation in the soil and long-term increases in the groundwater system may still occur. Effective seepage control through lagoon lining, lagoon relocation or other means should be exercised.

(c) Un-ionized Ammonia

Contributions of un-ionized ammonia ( $\text{NH}_3$ ) to Cookson Reservoir and the East Poplar River from mine dewatering were examined and it was concluded that contributions from that source should not be a concern. If the releases from the main boilerhouse sump at the Poplar River power plant are directed to a retention and treatment facility, additional small contributions from the boiler blowdown would, as well, be effectively controlled. The percent of ammonia present in the un-ionized form is strongly dependent on pH and temperature with the  $\text{NH}_3$  portion increasing sharply as pH increases above 8.0. Because the Cookson Reservoir water will have a pH in this range, build-up of ammonia levels over time should be prevented through ensuring adequate source control.

(d)

#### Flow Release for Habitat and Spawning Area Maintenance

The request from the Biology Committee to examine the possibility of a 20 m<sup>3</sup>/s release for 2 consecutive days each year was examined for effects on water availability in Cookson Reservoir. This release represents a loss from the reservoir of 3,456,000 m<sup>3</sup> per year or 8 percent of the full water volume in the reservoir. It is also more than the expected mean annual forced evaporation water loss for 2 units. A release of that magnitude and duration could not be permitted and still maintain power plant operation. In addition, under natural conditions the peak daily flow in the East Poplar River at the International Boundary was less than 20 m<sup>3</sup>/s for 24 of the 45 years of recorded peaks from 1931-76. There is one period of 5 and another of 4 consecutive years during which the peak flow did not reach 20 m<sup>3</sup>/s even for one day. Hence, the recommended flow release would not only preserve but would substantially enhance downstream habitat conditions. The Poplar River project should not be required to improve conditions. It is inappropriate to request releases higher than the peak reservoir inflow in any given year.

The minimum release-on-demand under the recommended flow apportionment of 370,000 m<sup>3</sup> would have a duration of only 5 hours if released at a rate of 20 m<sup>3</sup>/s. Hence, a very short term release at the 20 m<sup>3</sup>/s rate could be achieved if the downstream interests in containment for irrigation purposes find such a release mode acceptable. In any event, a flow release suitable for downstream habitat and spawning area maintenance should be achieved through adjustment of the discharge period for the release-on-demand volume recommended for flow apportionment.

It should also be noted that the maximum release rate possible when the reservoir level drops below the service spillway crest elevation of 750.87 m is less than 4 m<sup>3</sup>/s.

(e)

#### Nitrate and Total Phosphorus in Cookson Reservoir

The environmental assessment report (SPC, 1978) suggests that nitrate and phosphate levels in the reservoir may be of concern. Although the results have not been discussed earlier, N and P levels were computed for most simulation model runs. The computed reservoir concentrations for N (total) typically average about 4 mg/l and range from 0.5 to 6 mg/l whereas P (total) averages about 0.2 mg/l. The N levels are within the 10 mg/l recommended limit for the border. No border limit has been suggested for P, inferring that it is not considered to be a problem at the levels expected. The maximum P (total) level recorded for the East Poplar River at the International Boundary is 0.15 mg/l.

8.5

#### Recommended Mitigation Measures

The foregoing sections of this report suggest the mitigation measures listed below be adopted for the Poplar River power development.

(a)

### Ash Disposal System

No exact ash disposal system design is being recommended to leave some flexibility in the choice of the final system design. Instead, a number of design criteria are recommended that will ensure the effects of the ash disposal system on water quality in Cookson Reservoir and the East Poplar River are minimized. These design criteria are as follows:

- (i) if a lagoon system of ash disposal is adopted, any decant or blowdown releases to Cookson Reservoir should be treated to a reservoir water quality or better level before being released. It is recognized that the impracticality of treating decant from a once-through combined ash lagoon system may preclude serious consideration of that option.
- (ii) seepage from the ash lagoon or an ash disposal area to the East Poplar River downstream of Morrison Dam and to Cookson Reservoir should be minimized. Recommended maximum mean annual seepage rates for use in design of seepage control features are 2 l/s to the East Poplar River and 5 l/s to Cookson Reservoir from the total ash disposal area anytime during the life of the project.
- (iii) in determining seepage losses to the East Poplar River, the assumption should be made that the Empress Group formation has the capacity to readily convey seepage flows higher than the recommended limits. Seepage control measures should, therefore, ensure that rates of downward seepage through the lining and upper till layer are within the recommended limits.
- (iv) Lagoon lining, subdrainage provisions, seepage containment and treatment, lagoon relocation, change of disposal option or any other measures that can be demonstrated to keep seepage rates to the recommended levels would be acceptable forms of mitigation. Seepage pumpback should not be used unless it is clearly demonstrated that such a system would be effective without detrimentally affecting the local groundwater system or creating problems of pumpback water disposal.
- (v) a monitoring program should be established to determine the rates, destination and quality of seepage and discharge from the ash lagoon or ash disposal area to confirm compliance with the recommended maximum seepage rates and discharge water quality, and identify any water quality problems that may warrant further remedial action. If a lagoon disposal method is selected, it is recommended that a test area be established with a drainage and collection system beneath the selected lining.

Although a particular ash disposal system is not being recommended, a dry flyash disposal alternative is considered to be most effective in minimizing border water quality deterioration as well as reducing uncertainties with seepage rates and seepage control options. Should a lagoon option be selected, sufficient scientific information should be provided with the final design to demonstrate with reasonable assurance that the selected seepage control provisions will keep seepage rates within the recommended limits. This may require additional supportive laboratory and field experiments. If reasonable assurance that the recommended maximum seepage rates will be met cannot be provided, the lagoons should be relocated north of the plant and reservoir such that the recommended limits can be achieved. Further, a commitment should be made by the SPC to install more stringent control or adopt a dry flyash disposal option if monitoring during operation shows that lagoon seepage is not being effectively controlled.

(b) Supplemental Mitigation for TDS Reduction

It is recommended that monitoring of TDS levels in Cookson Reservoir be undertaken to determine whether the TDS level in the reservoir after spring freshet inflow each year is greater than 1000 mg/l. A commitment should be obtained from the SPC to take one of the following mitigative actions upon the first exceedance of the 1000 mg/l TDS concentration within a one month period after the spring peak flow.

- (i) install and operate a reservoir water softening facility to maintain TDS levels crossing the International Boundary within the recommended border limits. This could be accomplished by treating cooling water, mine dewatering discharge or both. If this supplemental option is preferred, space and design provisions to accommodate a softening facility should be provided and the facility should be constructed within one year of the first measured post-spring freshet TDS greater than 1000 mg/l.
- (ii) construct a storage and diversion facility on the Poplar River or on the East Poplar River upstream of Cookson Reservoir capable of containing and conveying sufficient good quality spring runoff water to maintain border TDS levels within the recommended limits by supplying or diluting either or both of the release-on-demand or low-flow discharges from Cookson Reservoir. Such facilities should be operational within 2 years of the first measured post-spring freshet TDS greater than 1000 mg/l.

(c) Containment and Treatment of Power Plant Wastestreams

All power plant wastestreams should be contained and treated to prescribed effluent limits before release to Cookson Reservoir. The following prescribed effluent limits are recommended:

pH	6.5 to 9.5	Ni	0.5 mg/l
TSS	25.0 mg/l	Cu	0.5 mg/l
Oil & Grease	15.0 mg/l	Zn	0.5 mg/l
Fe	1.0 mg/l	Cr	0.5 mg/l

In addition to treating to the prescribed limits for the elements listed, releases to the reservoir should be periodically monitored for the full range of elements normally identified as potential health hazards or having bioaccumulation potential in aquatic systems and food chains. Should any of these elements be identified as a potential problem from the results of monitoring, limitations for the problem elements should be added to the list of prescribed effluent limitations and treatment facilities should be improved to meet the additional requirements.

Regarding control of seepage from wastewater containment facilities, all ponds created for storage, equilization or evaporation of wastewaters should be located or lined to minimize seepage of poor quality water to the groundwater system. Design permeabilities should not be greater than  $10^{-6}$  cm/s for wastewater retention ponds. The containment facilities for transformer oil leaks or spills should be re-designed to provide full containment with no allowance for seepage into underlying sand as now proposed and measures should be taken to ensure that seepage from the sewage irrigation area does not enter the site drainage system.

To ensure that effective containment and treatment of power plant wastestreams is achieved, the proposed waste management system for the Poplar River generating station should be re-examined to ensure compatibility with the selected ash disposal system alternative. New containment facilities or alterations to those presently proposed may be required to accommodate wastewaters that could no longer be disposed of in the ash lagoons because of their contributing effect to operational problems or because of the adoption of dry flyash disposal. The SPC should provide, within a 2-month period of receipt of the IJC report to governments, an updated waste management system design compatible with the selected ash disposal system detailing all containment, treatment and seepage control features that are included to meet the above recommendations.

(d) Mining Operation

Efforts should be made to minimize the quantity of mine dewatering. Net supplies to Cookson Reservoir from mine dewatering improves reservoir water levels but, at an average TDS concentration of 1100 mg/l, contributes to concentrations exceeding the long-term recommended border TDS limit of 1000 mg/l. The control levels imposed by Environment Saskatchewan for suspended solids, turbidity and boron should adequately control the adverse effect of these parameters on Cookson Reservoir.

The practices and procedures described in Sections 3.6.3 and 3.6.4 should be adopted for control of effects of diverted runoff and non-point source discharges. A settling pond for retention of mine pit and sump drainage as now planned by the SPC should be employed for suspended solids control.

(e) Fife Lake Overflows

Fife Lake overflow would not likely occur unless there were several high precipitation years in succession. At that time, water quality in both Cookson Reservoir and Fife Lake should be better than average. Nevertheless, the overflow quality could still be considerably worse than Cookson Reservoir water quality and the overflow could be retained in the reservoir depending on the relative timing of the overflow and spillage at Morrison Dam. This retention in the system could in turn lead to poorer reservoir water quality. It is therefore recommended that facilities be provided to prevent future overflows from Fife Lake. Such facilities might be designed to permit planned releases to control flooding around the lake if it was determined that the quality of water in both Fife Lake and Cookson Reservoir were such that no significant deterioration in Cookson Reservoir quality would occur.

(f) Power Plant Operation

It is recommended that any planned power plant unit outages for maintenance or other reasons be made during the months of highest forced evaporation loss if there is a choice of when the outage can occur. This would improve water availability and reduce the concentrating effects caused by evaporation.

(g) Inspection of Pollution Control Features

A standing committee should be established to follow up on implementation of the Board's mitigation recommendations, with members from the water pollution control regulatory agencies in both Canada and the United States and from the SPC. The committee would (i) examine final design plans for the ash disposal and waste management systems to ensure they are consistent with the Board's recommendations and (b) inspect those systems during and upon completion of construction and during the initial period of plant operation. Such a group might best be composed of participants in the current IJC study who are familiar with the project.



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ANNEX A  
DETAILED MINING PLAN

Annex A contains estimated quantity data for the first 5 years of lignite mining (Block 1) at the Coronach Mine and a list of mining equipment to be used.



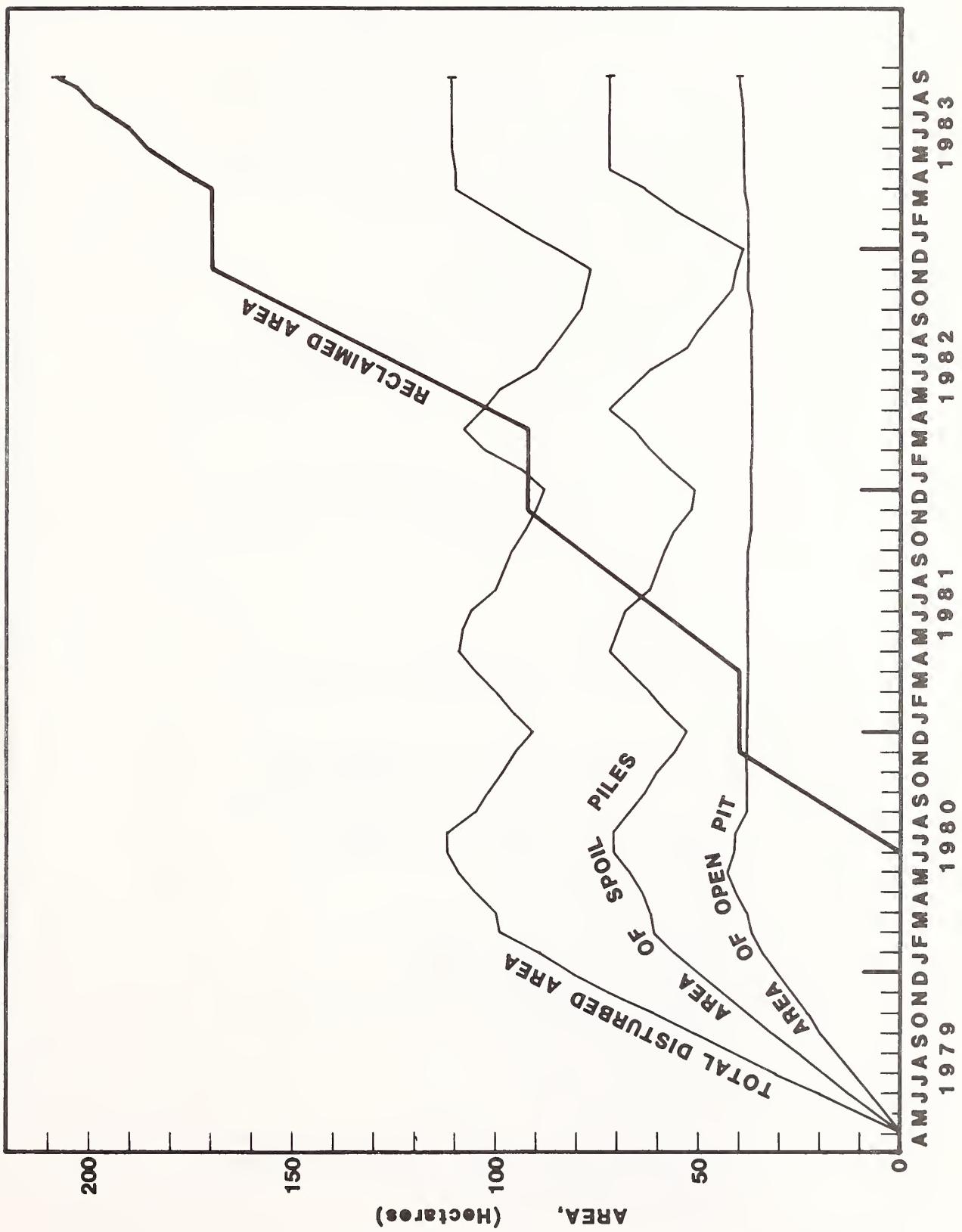


Figure A-1. Mine pit, spoil area, and reclaimed land area in mining block 1, Coronach mine.

Table 3-7. Mining Equipment and Mine Service Building Equipment List.

Mining Equipment List

1	Dragline, 2570-W Bucyrus-Erie
1	Power Shovel, 195-B Bucyrus-Erie
2	Locomotive, 750 kw (1000-horsepower)
15	Hopper Cars, 82-tonne (90-ton)
4	Coal Haulers, 136-tonne (150-ton) capacity
1	Mobile Crane, 91-tonne (100-ton)
1	All-Terrain Crane, 32-tonne (35-ton)
1	Articulated Dozer, 450 kw (600-horsepower)
1	Front-End Loader, 12 m <sup>3</sup> (16 yd. <sup>3</sup> )
1	Grader, 150 kw (200-horsepower)
1	Scraper, 27 m <sup>3</sup> (35 yd. <sup>3</sup> )
1	Float, 91-tonne (100-ton)
1	Pony, for 91-tonne float
1	Prime Mover (highway tractor)
1	Water Tanker, 26,500 l (7,000-gallon)
1	Fuel Truck, 11,350 l (3,000-gallon)
2	Service Trucks, 12,500 kg (27,500-pound) GVW
1	Cable Reel Carrier
1	Farm Tractor
3	Suburban Vans
6	Pickup Trucks, 2/3-tonne (3/4-ton)
3	Crawler Tractors, 300 kw (400-horsepower)
1	Crusher Feeder, Stamler
1	Coal Drill
1	Seed Press Drill
1	Rock Picker
1	Disc Harrow
1	Deep Tillage Cultivator
1	Farm Tractor, 75 kw (100-horsepower)

Mine Service Building Equipment List

1	Horizontal Milling Machine, heavy duty
1	Engine Lathe, 440-volt, 3-phase
1	Engine Lathe, 120-volt
1	Pedestal Grinder
1	Radial Drill
1	Drill Press, floor model
1	Drill Press, bench model
1	Horizontal Bandsaw
1	Vertical Hydraulic Press, 135-180 tonne
1	Power Hack Saw
1	Pedestal Grinder
1	Bolt Threader
1	Steam Cleaner
1	Air Compressor, 8.5 m <sup>3</sup> /minute (300 cfm)
1	Water Treatment Plant (including filters and chlorinator)
2	Welders, 600-ampere

**Table A-1. Detailed Mining Plan, Block 1, Coronach Mine.**

YEAR	MONTH	REQUIRED LIGNITE VOLUME (m³)	CUT NO.	ADVANCE IN CUT DURING MONTH (m)	TOTAL ADVANCE IN CUT (m)	CUT LENGTH REMAINING (m)	CHANGE IN PIT AREA (ha²)	TOTAL PIT AREA (ha²)	GENERATED SPOIL AREA (ha²)	TOTAL SPOIL AREA (ha²)	AREA RECLAIMED DURING MONTH (ha²)	TOTAL AREA RECLAIMED (ha²)	TOTAL AREA DISTURBED (ha²)	TOPSOIL STRIPPING ADVANCE (ha²)	TOTAL AREA STRIPPED OF TOPSOIL (ha²)	TOPSOIL STRIPPED IN MONTH (m³)	TOPSOIL STOCKPILED (m³)	VOLUME OF STOCKPILED TOPSOIL (m³)	SEEING WALL AREA (ha²)	MONTHLY VOLUME OF SEEPAGE (m³)
1979	April	51,650	1	440	440	3,840	3.88	3.88	6.35	6.35	0.00	0.00	10.23	13.31	13.31	50,700	50,700	2.48	9,000	
	May	51,650	1	430	870	3,410	3.88	7.76	6.35	12.70	0.00	0.00	20.46	13.31	26.62	50,700	101,400	4.79	17,900	
	June	51,650	1	440	1,310	2,970	3.88	11.64	6.35	19.05	0.00	0.00	30.69	13.31	39.93	50,700	152,100	7.11	25,700	
	July	51,650	1	430	1,740	2,540	3.88	15.52	6.35	25.40	0.00	0.00	40.92	13.31	53.24	50,700	202,800	9.42	35,200	
	August	51,560	1	440	2,180	2,100	3.88	19.40	6.35	31.75	0.00	0.00	51.15	13.31	66.55	50,700	253,500	11.74	43,900	
	September	51,560	1	440	2,620	1,660	3.88	23.28	6.35	38.10	0.00	0.00	61.38	13.30	78.85	50,700	304,200	14.05	50,800	
	October	51,560	1	440	3,050	1,230	3.88	27.16	6.35	44.45	0.00	0.00	71.61	13.30	93.15	50,700	354,900	16.37	61,200	
	November	45,200	1	380	3,430	850	3.39	30.55	5.55	50.00	0.00	0.00	80.55	12.03	105.18	45,800	400,700	18.40	66,600	
	December	46,700	1	390	3,820	460	3.51	34.06	5.74	55.74	0.00	0.00	89.80	0.00	105.18	0	400,700	20.49	76,600	
1980	January	46,700	1	390	4,210	70	3.51	37.57	5.75	61.49	0.00	0.00	99.05	0.00	105.18	0	400,700	22.58	84,400	
	February	43,690	2	320	320	2,160	0.76	38.32	0.68	62.17	0.00	0.00	100.50	0.00	105.18	0	400,700	22.16	77,500	
	March	162,520	2	1,200	1,520	960	2.57	40.90	2.11	64.28	0.00	0.00	105.18	0.00	105.18	0	400,700	19.40	72,500	
	April	157,280	3	200	200	2,250	1.82	42.72	2.34	66.62	0.00	0.00	109.34	6.46	111.64	24,600	425,300	17.18	62,100	
	May	162,520	3	1,220	1,420	1,030	-1.55	41.17	4.96	70.58	0.00	0.00	111.75	5.38	117.02	20,500	445,800	17.18	64,200	
	June	0	3	0	1,420	1,030	0.00	41.17	0.00	70.58	0.00	0.00	111.75	5.37	122.39	20,500	466,300	17.2	62,100	
	July	162,520	4	220	220	2,000	-3.05	38.12	4.08	66.70	7.96	7.96	104.82	5.38	127.77	20,500	462,500	17.2	64,200	
	August	162,520	4	1,240	1,460	760	0.13	38.25	4.94	63.68	7.96	15.92	101.93	5.37	133.14	20,500	458,800	17.2	64,200	
	September	157,280	5	440	440	1,720	0.05	38.30	4.13	59.85	7.96	23.88	98.15	5.38	136.52	20,500	455,000	17.2	62,100	
	October	162,520	5	1,250	1,690	470	-0.09	38.21	4.46	56.35	7.96	31.84	94.56	5.37	143.89	20,500	451,200	17.2	64,200	
	November	157,280	6	780	780	1,260	-0.03	38.18	4.49	52.88	7.96	39.80	91.06	5.38	149.27	20,500	447,400	17.2	62,100	
	December	162,520	7	10	10	1,960	0.00	38.18	4.76	57.64	0.00	39.80	95.82	0.00	149.27	0	447,400	17.2	64,200	
1981	January	162,520	7	1,290	1,300	670	-0.11	38.07	4.73	62.37	0.00	39.80	100.44	0.00	149.27	0	447,400	17.2	64,300	
	February	146,800	8	520	520	1,340	-0.09	37.98	4.32	66.69	0.00	39.80	104.67	0.00	149.27	0	447,400	17.2	58,100	
	March	162,520	8	1,300	1,820	40	-0.08	37.90	4.88	71.57	0.00	39.80	109.47	0.00	149.27	0	447,400	16.6	62,100	
	April	157,280	9	1,240	1,240	510	0.04	37.94	4.73	69.78	6.52	46.32	107.72	7.27	156.54	27,700	455,300	16.6	60,000	
	May	162,520	10	820	820	850	-0.07	37.87	4.89	68.16	6.51	52.83	106.03	7.32	163.86	27,800	463,300	16.0	59,800	
	June	0	10	0	820	850	0.00	37.87	0.00	61.64	6.52	59.35	99.51	6.06	169.92	23,100	466,500	16.0	57,900	
	July	162,520	11	440	440	1,090	-0.09	37.78	4.75	59.89	6.51	65.86	97.67	6.07	175.99	23,100	469,800	15.4	57,600	
	August	162,520	12	130	130	2,570	-0.04	37.74	4.55	57.92	6.52	72.38	95.66	6.06	182.05	23,100	473,000	14.8	55,300	
	September	157,280	12	1,300	1,430	1,270	-0.05	37.64	4.73	56.14	6.51	78.89	93.78	6.07	188.12	23,100	476,300	14.2	51,400	
	October	162,520	13	80	80	2,640	-0.71	36.93	3.16	52.78	6.52	85.41	89.71	6.06	194.18	23,100	479,500	13.6	50,800	
	November	157,280	13	1,280	1,360	1,360	-0.03	36.90	5.21	51.47	6.52	91.93	88.37	6.07	200.23	23,100	482,800	13.6	49,200	
	December	162,520	14	10	10	2,720	-0.04	36.86	5.38	56.85	0.00	169.93	93.71	0.00	200.25	0	482,800	13.6	50,800	
1982	January	162,520	14	1,370	1,380	1,350	-0.03	36.83	5.06	61.91	0.00	91.93	98.74	0.00	200.25	0	482,800	13.6	50,800	
	February	146,800	14	1,240	2,620	110	-0.03	36.80	4.56	66.47	0.00	91.93	103.27	0.00	200.25	0	482,800	13.6	45,900	
	March	162,520	15	1,300	1,300	1,460	-0.04	36.76	5.08	71.55	0.00	91.93	108.31	0.00	200.25	0	482,800	13.6	50,800	
	April	157,280	15	1,330	2,630	130	-0.03	36.73	4.92	66.72	9.75	101.68	103.45	9.98	210.23	38,000	491,000	13.6	49,200	
	May	162,520	16	1,210	1,210	1,550	-0.02	36.71	4.84	61.81	9.75	111.43	98.52	9.97	220.20	38,000	499,300	13.6	50,800	
	June	45,200	16	360	1,570	1,190	-0.01	36.70	1.35	53.41	9.75	121.18	90.11	9.98	230.18	38,000	507,600	13.6	49,200	
	July	209,230	17	480	480	2,280	0.07	36.77	6.16	49.82	9.75	130.93	86.59	9.97	240.15	38,000	515,900	13.6	50,800	
	August	209,230	17	1,640	2,120	640	0.32	37.09	6.04	46.11	9.75	140.68	83.20	9.98	250.13	38,000	524,200	13.6	50,800	
	September	202,480	18	900	900	1,880	0.22	37.31	5.56	41.92	9.75	150.43	79.23	9.97	260.10	38,000	532,400	13.6	49,200	
	October	325,050	19	570	570	2,220	0.24	37.55	8.59	40.76	9.75	160.18	78.31	9.98	270.08	38,000	540,700	13.6	50,800	
	November	314,560	20	40	40	2,770	0.22	37.77	8.12	39.13	9.75	169.93	76.90	9.98	280.06	38,000	549,000	13.6	49,200	
	December	325,050	20	2,360	2,400	410	0.18	37.95	8.40	47.53	0.00	169.93	85.48	0.00	280.06	0	549,000	13.6	50,800	
1983	January	325,050	21	1,890	1,890	910	0.24	38.19	8.29	55.82	0.00	169.93	94.01	0.00	280.06	0	549,000	13.6	50,800	
	February	293,590	22	1,140	1,140	1,700	0.17	38.36	7.46	63.28	0.00	169.93	101.64	0.00	280.06	0	549,000	13.6	45,900	
	March	325,050	23	640	640	2,150	0.19	38.55	8.30	71.58	0.00	169.93	110.13	0.00	280.06	0	549,000	13.6	50,900	
	April	314,560	24	110	110	2,710	0.24	38.79	8.12	71.58	8.12	178.05	110.37	8.36	288.42	31,900	556,100	13.6	49,200	
	May	325,050	24	2,530	2,640	180	0.45	39.24	8.36	71.59	8.35	186.40	110.83	8.81	297.53	33,600	564,200	13.6	50,800	
	June	157,280	25	780	780	2,100	-0.01	39.23	4.05	71.59	4.05	190.45	110.82	4.04	311.27	15,400	567,300	13.6	49,200	
	July	325,050	26	310	310	2,600	0.23	39.46	8.41											



## ANNEX B

### PREPARATION, DISTRIBUTION AND ANALYSIS OF POPLAR RIVER LIGNITE AND ASH SAMPLES

Annex B contains the following:

- (i) A description of the method of preparing lignite and ash samples from the Coronach mine area.
- (ii) Seven summary tables containing the results of analyses conducted on Poplar River lignite and ash samples.



## PREPARATION OF LIGNITE AND ASH SAMPLES

Many analyses of lignite and ash from the Coronach mine area have been reported; however, there still remains the question of how well such analyses represent the lignite that will be used to fuel the Coronach powerplant. In addition, studies of trace element content and distribution in the lignite and ash are required.

In order to perform these studies, a number of samples of lignite and ash from the test burn of a 500-ton sample of Coronach mine lignite were requested. This report describes the methods by which representative portions of each sample were produced and the manner in which samples were mixed to produce representative composites of the originally provided materials.

### Poplar River lignite samples from B.D.P.S. Unit /1 test burn

The Saskatchewan Power Corporation provided the International Poplar River Water Quality Board Committee for Plant, Mine, and Reservoir Operations with 7 samples of dried and pulverized lignite from the Coronach mine. These samples were taken during a 500-ton test burn of Coronach lignite at the Boundary Dam Power Station Unit /1 on December 19, 1975. The following weights of samples were furnished:

1 A Mill 0300	47.6 grams
1 D Mill 0400	187.9 grams
1 B Mill 0400	167.8 grams
1 A Mill 0500	127.6 grams
P.R. Coal 0500	140.8 grams
1 B Mill 0600	119.6 grams
1 D Mill 0600	184.6 grams

After weighing, each sample was mixed by repeated rolling and coning on a sheet of smooth paper. The sample was then passed through a sample splitter and each of the two splits was again mixed and passed through a sample splitter a second time, resulting in 4 aliquots of the original sample. Each portion was weighed and one portion was labeled and placed in an envelope while each of the remaining 3 portions was placed on a mixing sheet and composited with respective portions of each of the other 6 lignite samples. There resulted from this compositing procedure 3 lignite samples containing the following weights of the originally provided material:

<u>Original lignite sample</u>	<u>Composite /1</u>	<u>Composite /2</u>	<u>Composite /3</u>
1 A Mill 0300	11.8g	11.7g	11.7g
1 D Mill 0400	46.1g	51.0g	41.8g
1 B Mill 0400	41.0g	40.3g	42.0g
1 A Mill 0500	31.6g	30.8g	32.7g
P.R. Coal 0500	23.2g	39.3g	46.3g
1 B Mill 0600	30.5g	28.8g	29.5g
1 D Mill 0600	44.9g	45.8g	44.1g

Each composite sample was thoroughly mixed and split into two portions. In turn, each portion was mixed and again split into two portions which were then weighed and placed into labeled envelopes. The entire lignite sample preparation is shown in schematic form in figure B-1.

Bottom ash samples from B.D.P.S. Unit /1 test burn

Three samples of bottom ash aggregating 1.8 kg were provided by the Saskatchewan Power Corporation:

Bottom Ash	0500	469.2 grams
Bottom Ash	0545	512.2 grams
Bottom Ash	0650	869.4 grams

These samples were collected wet during the December 19, 1975, test burn of 500 tons of Coronach lignite at the Boundary Dam Power Station Unit /1. Although the bottom ash was collected wet, there were no samples of decant water collected and no analysis of the slurry water was attempted. No lime was added to the lignite during the test burn when these ash samples were collected.

After weighing, each sample of bottom ash was thoroughly mixed by repeated rolling and coning on a sheet of smooth paper. The mixed material was then separated into 2 portions by passing it through a sample splitter. Each portion was then mixed and again divided into 2 portions so that each original sample had been divided into 8 representative portions. Four of these portions were weighed and placed directly into labeled envelopes and the remaining 4 portions were weighed and added to the respective portions of each of the other original samples to yield 4 composites of the 3 original samples.

<u>Original Sample</u>	<u>Composite /1</u>	<u>Composite /2</u>	<u>Composite /3</u>	<u>Composite /4</u>
Bottom Ash 0500	57.0g	57.2g	60.1g	57.6g
Bottom Ash 0545	64.6g	66.4g	63.1g	61.3g
Bottom Ash 0650	109.3g	105.3g	108.9g	107.0g

Each composite sample was thoroughly mixed and split into 2 portions. Each portion was subsequently mixed and split into 2 portions, yielding 4 representative aliquots from each of the 4 composited bottom ash samples. Each aliquot was weighed and placed into a labeled envelope. Figure B-2 schematically shows the procedure used to composite and divide the 3 original bottom ash samples provided.

### Upper ash samples from B.D.P.S. Unit /1 test burn

Three samples of upper ash from a 500-ton test burn of Coronach mine lignite (without lime addition) in Unit /1 of the Boundary Dam Power Station were provided by the Saskatchewan Power Corporation.

Upper Ash 0410 184.1 grams  
Upper Ash 0500 320.4 grams  
Upper Ash 0650 179.5 grams

Each sample was weighed, then was thoroughly mixed by rolling and coning on a sheet of smooth paper and was separated into 2 portions by passing the material through a sample splitter. Each portion was again mixed and was further divided by passing it through the sample splitter a second time. The resulting 4 portions of each original sample were weighed and one portion was placed in a labeled envelope. The remaining 3 portions were added to the respective splits of each of the other original samples to produce 3 composited samples of the original 3 upper ash samples.

<u>Original sample</u>	<u>Composite /1</u>	<u>Composite /2</u>	<u>Composite /3</u>
Upper Ash 0410	48.4g	42.2g	45.6g
Upper Ash 0500	80.4g	79.5g	82.9g
Upper Ash 0650	40.2g	48.5g	52.0g

The composited samples each were thoroughly mixed and split into 2 portions, then each portion was mixed and again split into 2 portions. All portions were weighed and placed in labeled envelopes. A schematic diagram of this splitting and compositing procedure is shown in figure B-3.

### Dust collector ash samples from B.D.P.S. Unit /1 test burn

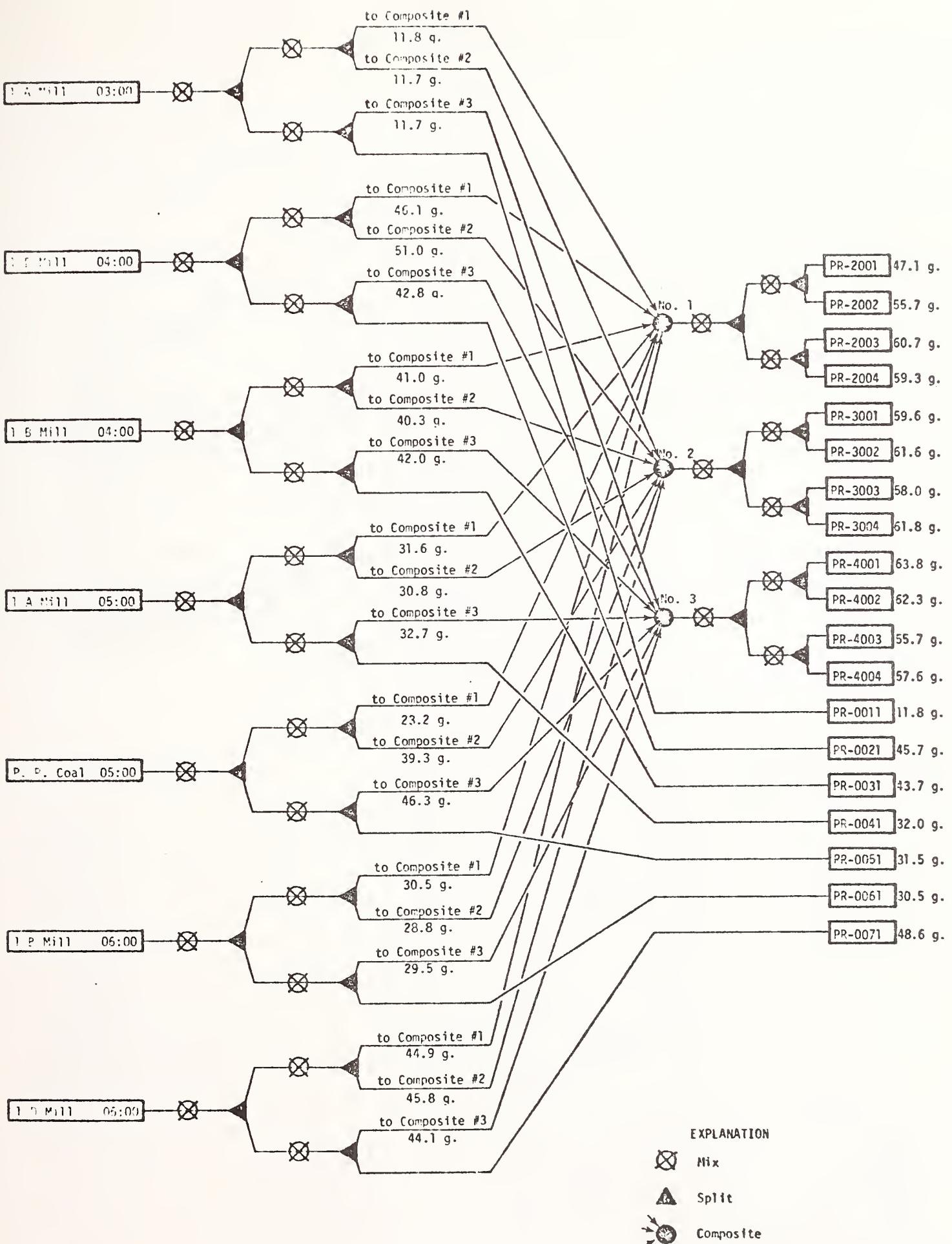
Four samples of ash obtained from the dust collector cyclone at the Boundary Dam Power Station Unit /1 during a 500-ton test burn of Coronach mine lignite (without lime addition) were provided by the Saskatchewan Power Corporation.

Dust Collector Ash 0310            212.7 grams  
Dust Collector Ash 0435            356.4 grams  
Dust Collector Ash 0540            344.5 grams  
Dust Collector Ash 0640            374.4 grams

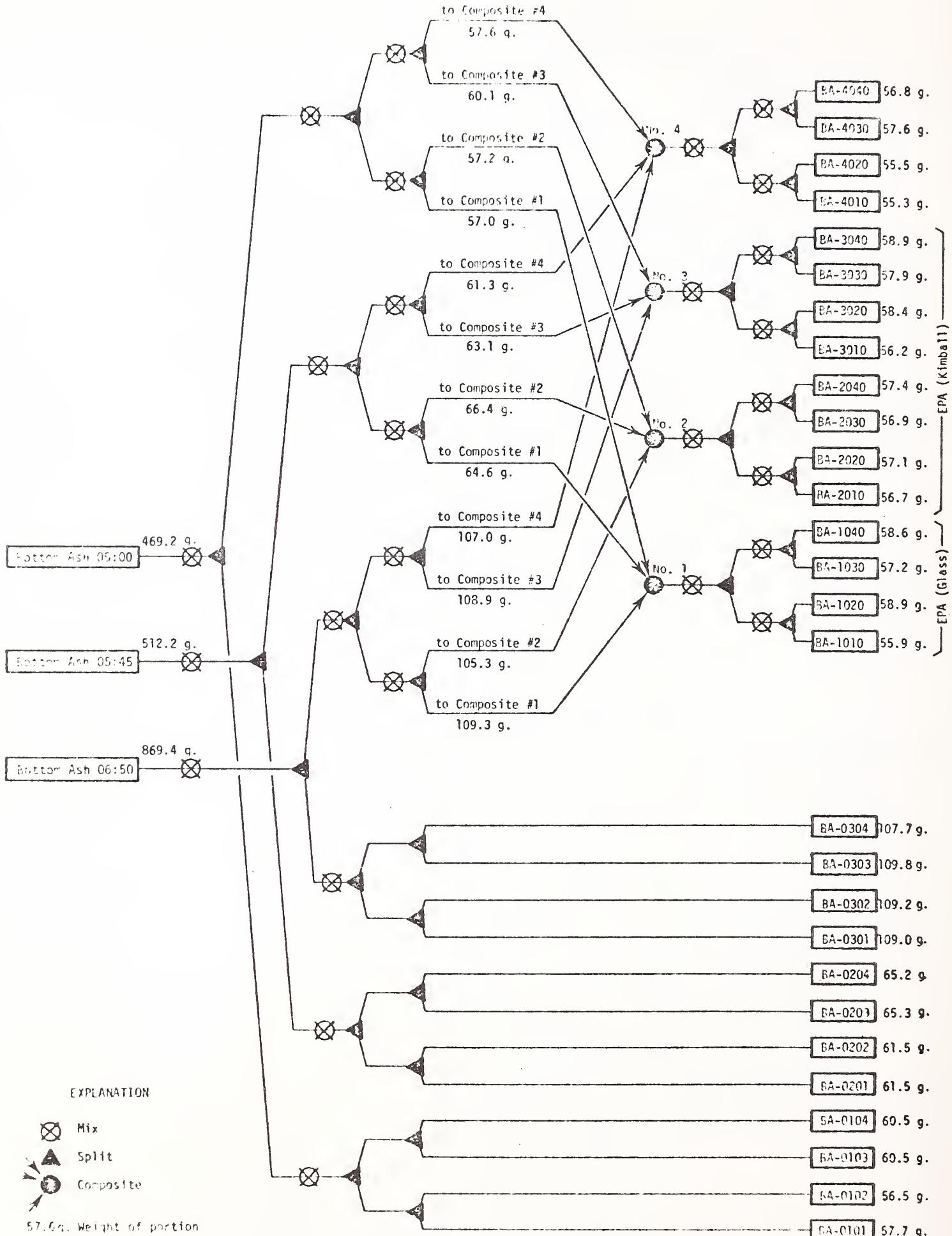
Each dust collector ash sample was weighed and thoroughly mixed by repeated rolling and coning on a sheet of smooth paper. The mixed material was divided into 2 portions by passing it through a sample splitter. These portions were individually mixed and again split into 2 portions. Each portion was weighed and one portion was placed in a labeled envelope. The remaining 3 portions were composited with respective portions of each of the other dust collector ash samples to yield 3 composites.

<u>Original Sample</u>	<u>Composite /1</u>	<u>Composite /2</u>	<u>Composite /3</u>
Dust Collector Ash 0310	50.6g	56.4g	53.0g
Dust Collector Ash 0435	91.3g	89.3g	88.3g
Dust Collector Ash 0540	94.1g	89.1g	84.7g
Dust Collector Ash 0640	99.9g	86.5g	88.7g

Each composite was thoroughly mixed and separated into 2 portions by passing through a sample splitter each portion was subsequently mixed and split into 2 parts. The 4 aliquot portions resulting from each composited sample were individually weighed and placed into labeled envelopes. A schematic representation of this compositing and splitting procedure is shown in figure B-4.

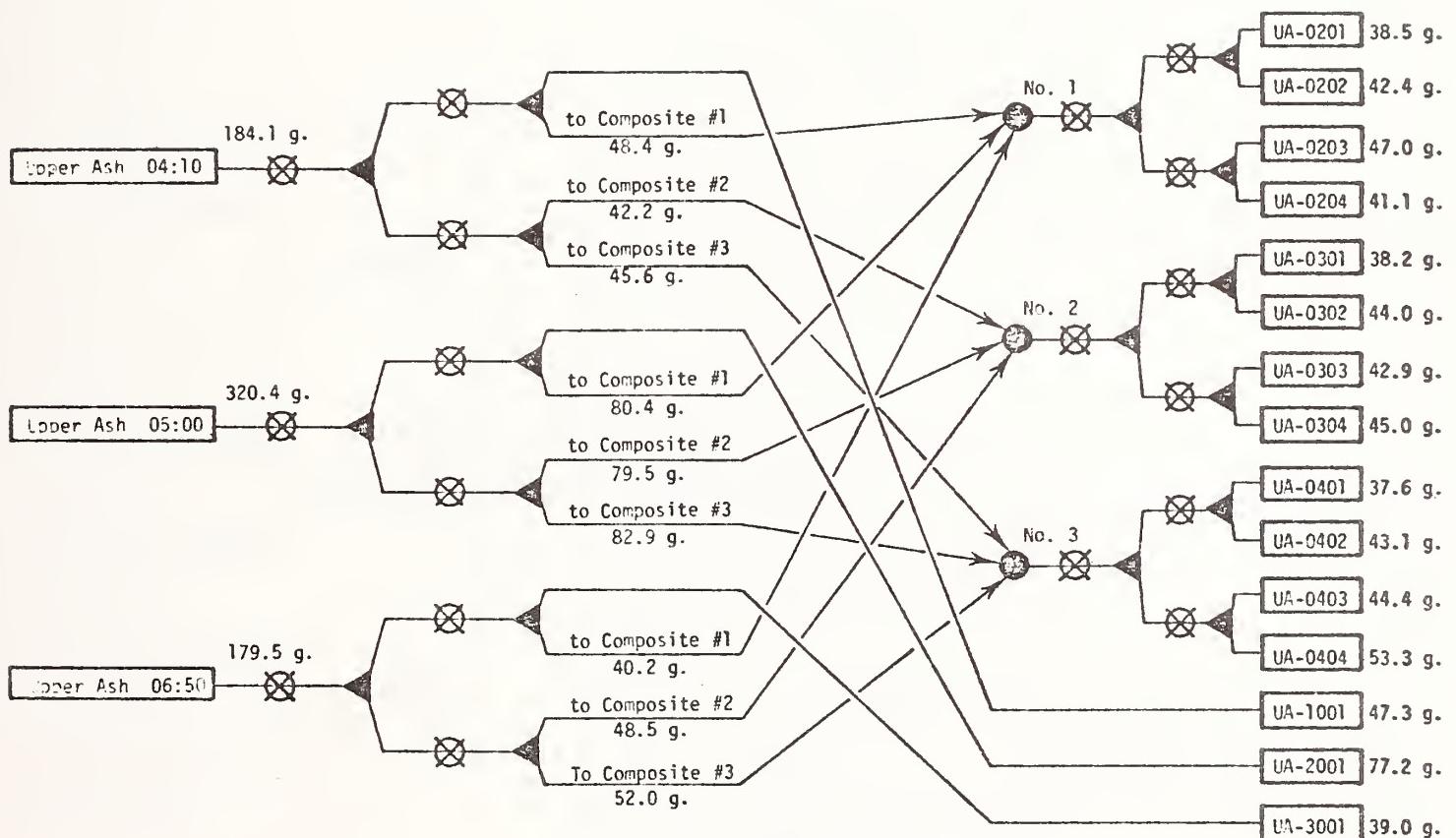


**FIGURE B-1**  
Schematic procedure for splitting and compositing  
coal samples from Poplar River coal test burn.



**FIGURE B-2**

Schematic procedure for splitting and compositing bottom ash samples from Poplar River coal test burn.



#### EXPLANATION

Mix

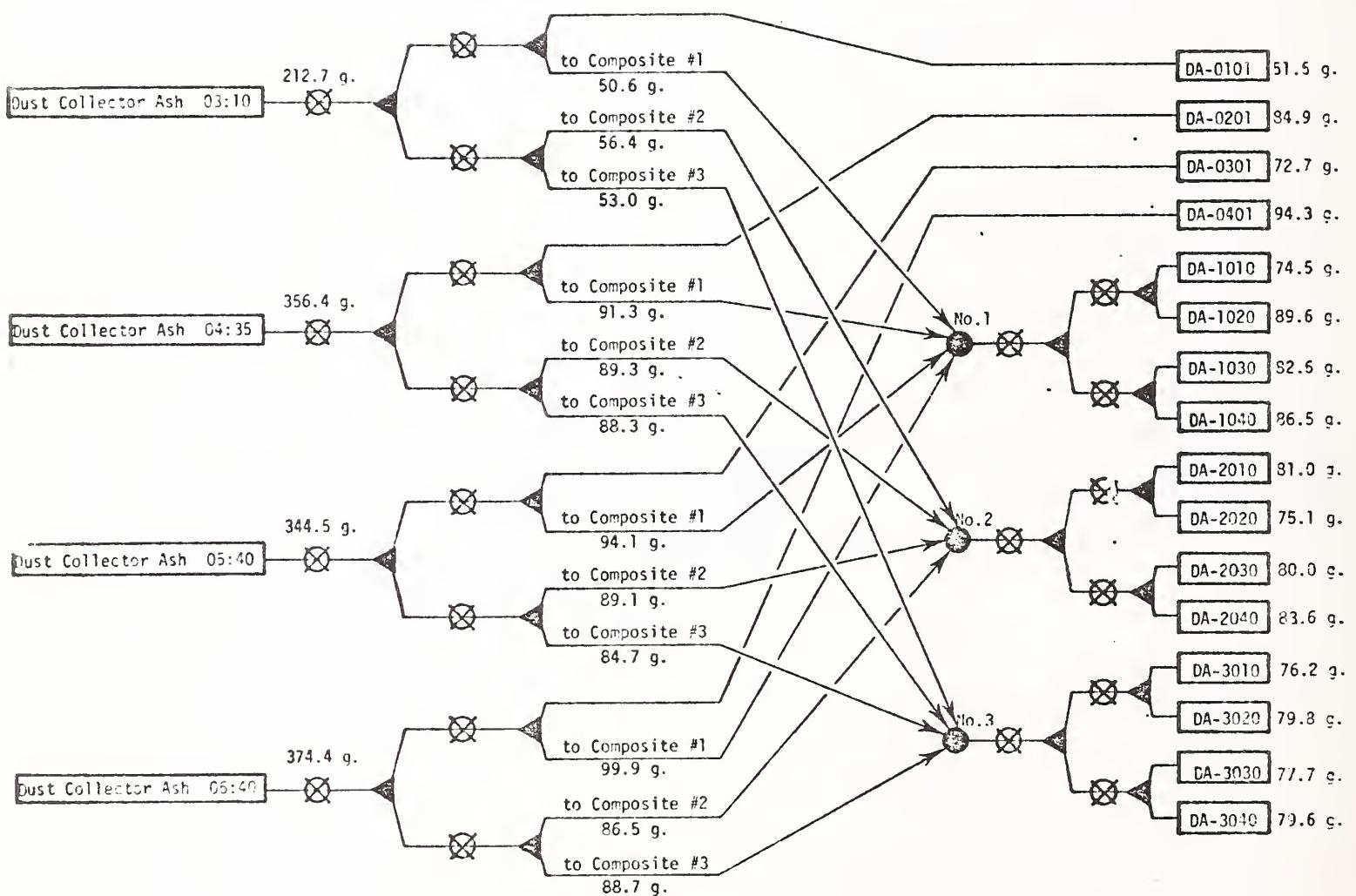
Split

Composite

179.5 g. Weight of portion

FIGURE B-3

Schematic procedure for splitting and compositing upper ash samples from Poplar River coal test burn.



#### EXPLANATION

Mix

Split

Composite

374.4 g. Weight of portion

FIGURE B-4

Schematic procedure for splitting and compositing dust collector ash samples from Poplar River coal test burn.

RESULTS OF ANALYSIS OF POPLAR RIVER  
LIGNITE AND ASH SAMPLES

TABLE B-1. Hart Seam Lignite, Minor and Trace Element Analyses.

Sample No.	Laboratory Code	Silicon %	Aluminum %	Calcium %	Magnesium %	Sodium %	Potassium %	Iron %	Titanium %
Rockglen 169	1	8.0	1.60	1.8	0.40	0.56	0.44	1.64	0.060
Rockglen 171	1	4.4	1.20	1.6	0.36	0.28	0.24	0.52	0.042
Rockglen 176	1	6.6	1.75	1.8	0.52	0.32	0.32	0.60	0.054
Rockglen 178	1	5.6	1.74	2.2	0.56	0.32	0.28	0.72	0.060
Rockglen 248	1	3.8	1.73	2.7	0.48	0.24	0.16	0.64	0.048
Rockglen 249	1	4.4	1.32	1.7	0.36	0.20	0.20	0.76	0.048
Rockglen 253	1	8.4	2.60	2.2	0.60	0.20	0.50	0.72	0.090
Rockglen 257	1	8.6	2.20	2.5	0.60	0.20	0.34	0.72	0.084
Rockglen 169	2		0.36*	1.60	0.341	0.347	0.59	0.64	0.0198
Rockglen 171	2		0.44*	1.40	0.286	0.394	0.57	0.27	0.0177
Rockglen 176	2		0.49*	1.60	0.408	0.327	0.64	0.53	0.0189
Rockglen 178	2		0.51*	1.99	0.503	0.377	0.75	0.43	0.0238
Rockglen 248	2		0.41*	1.60	0.428	0.269	0.56	0.33	0.0117
Rockglen 249	2		0.48*	1.40	0.396	0.340	0.58	0.90	0.0164
Rockglen 253	2		0.50*	1.60	0.404	0.333	0.71	0.38	0.0151
Rockglen 257	2		0.42*	1.78	0.412	0.363	0.72	0.43	0.0234
Rockglen 169	2	3.58	1.97	2.01	0.52	0.24	0.36	0.84	
Rockglen 171	2	1.92	1.32	1.61	0.38	0.16	0.20	0.42	
Rockglen 176	2	2.70	1.68	2.02	0.48	0.14	0.29	0.68	
Rockglen 178	2	2.32	1.58	2.09	0.50	0.15	0.28	0.51	
Rockglen 248	2	1.71	1.30	1.77	0.48	0.15	0.17	0.43	
Rockglen 249	2	1.81	1.31	1.76	0.43	0.11	0.20	0.59	
Rockglen 253	2	3.55	2.14	1.90	0.53	0.18	0.41	0.57	
Rockglen 257	2	3.69	2.23	2.25	0.55	0.21	0.38	0.66	
Willow Bunch 169	3	5.6		2.0		0.18			
Willow Bunch 171	3	2.1		1.5		0.28			
Willow Bunch 176	3	3.8		1.8		0.12			
Willow Bunch 178	3	3.0		1.7		0.12			
Willow Bunch 248	3	1.7		1.4		0.11			
Willow Bunch 249	3	1.8		1.3		0.09			
Willow Bunch 253	3	4.6		1.5		0.26			
Willow Bunch 257	3	4.1		1.7		0.31			
Poplar River #1	4								
Poplar River #5	4								
PR 2001	5		1.92	2.09	0.54			0.66	
PR 2002	6A	3	1.1	1	0.10	0.02	0.11	0.47	0.0900
PR 2002	6B		2.1	1.5	0.60	0.18	0.36	0.74	0.013
PR 4001, 4002	4								
Arithmetic Mean (No. of Samples)		4.0 (9)	1.7 (10)	1.8 (10)	0.45 (10)	0.23 (9)	0.39 (9)	0.62 (10)	0.051 (9)

\*Numerical value probably in error, not used in computation of mean.

TABLE B-1(cont.). Hart Seam Lignite, Minor and Trace Element Analyses.

Sample No.	Laboratory Code	Manganese ppm	Nickel ppm	Chromium ppm	Cobalt ppm	Lithium ppm	Strontium ppm	Copper ppm	Zinc ppm	Lead ppm
Rockglen 169	1	160	40	10	30	8.1	80	180	120	50
Rockglen 171	1	160	80	10	10	4.9	120	180	50	40
Rockglen 176	1	150	40	15	10	6.6	80	180	40	120
Rockglen 178	1	150	20	10	10	6.0	80	110	50	100
Rockglen 248	1	140	40	20	10	4.8	80	70	90	80
Rockglen 249	1	140	120	20	20	4.9	80	70	60	80
Rockglen 253	1	140	250	30	20	9.2	60	60	20	90
Rockglen 257	1	230	130	40	10	14.6	60	40	90	100
Rockglen 169	2	137	59.0	20.5	6.9		174	80.0	990*	25
Rockglen 171	2	145	78.7	19.2	5.7		193	120	1550*	20
Rockglen 176	2	109	73.7	33.2	8.3		167	91.6	1220*	12
Rockglen 178	2	130	86.4	30.9	10.5		217	103	1450*	13
Rockglen 248	2	84	57.4	15.7	7.5		145	72.3	1280*	12
Rockglen 249	2	199	85.6	26.0	8.3		165	118	1790*	16
Rockglen 253	2	100	74.6	22.0	11.0		192	94.8	1390*	12
Rockglen 257	2	193	71.0	22.0	9.8		213	94.8	1190*	14
Rockglen 169	2	143	10	48				24	24	22
Rockglen 171	2	151	4	25				15	19	18
Rockglen 176	2	101	15	44				19	17	<17
Rockglen 178	2	97	9	36				22	14	<16
Rockglen 248	2	119	6	25				14	25	18
Rockglen 249	2	123	5	34				17	13	19
Rockglen 253	2	96	6	37				20	24	<20
Rockglen 257	2	223	8	38				25	17	<21
Willow Bunch 169	3	141	18	25		11	209	13	15	20
Willow Bunch 171	3	134	14	18		7	188	12	13	24
Willow Bunch 176	3	103	22	31		10	140	14	9	15
Willow Bunch 178	3	105	15	27		9	120	16	7	11
Willow Bunch 248	3	100	10	12		8	119	11	7	21
Willow Bunch 249	3	106	9	8		8	139	13	10	20
Willow Bunch 253	3	100	15	18		14	149	16	16	27
Willow Bunch 257	3	204	22	32		22	120	18	14	20
Poplar River #1	4	44	2.8	2.9	2.4		240	16	330	10
Poplar River #5	4	240	15	6.7	1.6		280	9.0	2.8	10
PR 2001	5	118	<50	29			94	29		
PR 2002	6A	130	20	27	8			10		1
PR 2002	6B	140		29	1.4		270		14	
PR 4001, 4002	4	60	3.2	34	1.8		440	16		40
Arithmetic Mean (No. of Samples)		130 (12)	37 (11)	25 (12)	9.3 (11)	9 (8)	197 (12)	46 (12)	40 (11)	<29 (11)

\*Numerical value probably in error, not used in computation of mean.

TABLE B-1 (cont.). Hart Seam Lignite, Minor and Trace Element Analyses.

Sample No.	Laboratory Code	Cadmium ppm	Mercury ppm	Vanadium ppm	Uranium ppm	Fluorine ppm	Boron ppm	Phosphorus %	Arsenic ppm	Selenium ppm
Rockglen 169	1	40*			17	5403*				
Rockglen 171	1	0*			4	6053*				
Rockglen 176	1	10*			11	3614*				
Rockglen 178	1	20*			16	4846*				
Rockglen 248	1	20*			33	6914*				
Rockglen 249	1	10*			23	4776*				
Rockglen 253	1	10*			16	5227*				
Rockglen 257	1	10*			13	4430*				
Rockglen 169	2	ND	42*	20.5	ND	ND	0.131	ND	ND	
Rockglen 171	2	ND	20*	22.8	ND	ND	0.180	ND	ND	
Rockglen 176	2	ND	30*	20.9	ND	ND	0.130	ND	ND	
Rockglen 178	2	ND	30*	21.8	ND	ND	0.147	ND	ND	
Rockglen 248	2	ND	28*	19.3	ND	ND	0.186	ND	ND	
Rockglen 249	2	ND	30*	25.5	ND	ND	0.148	ND	ND	
Rockglen 253	2	ND	26*	23.6	ND	ND	0.143	ND	ND	
Rockglen 257	2	ND	42*	19.2	ND	ND	0.152	ND	ND	
Rockglen 169	2	<0.8	0.11		7.7	4.6		6.0	1.7	
Rockglen 171	2	<0.5	0.11		2.4	4.2		9.3	1.4	
Rockglen 176	2	<0.7	0.13		3.9	4.5		6.3	1.2	
Rockglen 178	2	<0.6	0.18		4.5	4.1		7.6	1.8	
Rockglen 248	2	<0.5	0.13		3.3	4.0		7.1	0.9	
Rockglen 249	2	<0.5	0.14		3.2	4.5		4.4	1.2	
Rockglen 253	2	<0.8	0.49		3.4	4.3		15.6	1.3	
Rockglen 257	2	<0.8	0.14		2.6	4.6		8.7	1.5	
Willow Bunch 169	3		0.54		116.0			6.0	0.2	
Willow Bunch 171	3		0.53		92.0			6.5	0.5	
Willow Bunch 176	3		0.70		95.0			8.0	0.2	
Willow Bunch 178	3		0.73		92.0			6.5	0.5	
Willow Bunch 248	3		0.36		62.0			5.5	0.0	
Willow Bunch 249	3		0.39		95.0			6.0	0.8	
Willow Bunch 253	3		0.36		123.0			15.0	0.0	
Willow Bunch 257	3		0.38		113.0			7.5	0.2	
Poplar River #1	4	.33	0.33	23	0.65	22	210		1.5	<0.29
Poplar River #5	4	.44	0.33	35	1.2	22	57		3.0	0.83
PR 2001	5	<10								
PR 2002	6A	ND			1	22	130		13	
PR 2002	6B			20	2.3		244		4.5	1.5
PR 4001, 4002	4	0.19	0.12	13	2.8	110	55		0.98	0.48
Arithmetic Mean (No. of Samples)		<0.6 (10)	0.32 (10)	21 (11)	7.9 (11)	86 (11)	37 (11)	0.152 (8)	6.8 (11)	0.8 (11)

\*Numerical value probably in error, not used in computation of mean.

ND = Not Detected

TABLE B-1 (cont.). Hart Seam Lignite, Minor and Trace Element Analyses.

Sample No.	Laboratory Code	Antimony ppm	Barium ppm	Beryllium ppm	Bismuth ppm	Bromine ppm	Molybdenum ppm	Thallium ppm	Zirconium ppm
Willow Bunch 169	3	ND		ND					
Willow Bunch 171	3	ND		ND					
Willow Bunch 176	3	ND		ND					
Willow Bunch 178	3	ND		ND					
Willow Bunch 248	3	ND		ND					
Willow Bunch 249	3	ND		ND					
Willow Bunch 253	3	ND		ND					
Willow Bunch 257	3	ND		ND					
Poplar River #1	4	2.2	410	0.19		0.96		0.72	
Poplar River #5	4	2.2	410	<0.15		2.2		<0.64	
PR 2001	5								35
PR 2002	6A		500		9	9	10	ND	140
PR 2002	6B	0.55	770			1.3	7.2		110
PR 4001, 4002	4	0.56	350	<0.10		1.4		0.38	
Arithmetic Mean (No. of Samples)		1.1 (3)	465 (3)	<0.14 (2)	9 (1)	2.7 (3)	9 (1)	<0.5 (2)	80 (2)

## Laboratory Codes

1. A. A. Detectametal Laboratories Ltd., Calgary, Alberta
2. Saskatchewan Research Council, Saskatoon, Saskatchewan  
(Mercury analyses by Barringer Research Ltd., Rexdale, Ontario)
3. Montana Department of Health and Environmental Sciences, Helena, Montana
4. CDM/Accu-Labs, Wheat Ridge, Colorado
5. U.S. Environmental Protection Agency, Analytical Support Division, Corvallis, Oregon
- 6A. U.S. Environmental Protection Agency, Environmental Research Laboratory, Athens, Georgia  
Spark Source Mass Spectrometry
- 6B. U.S. Environmental Protection Agency, Environmental Research Laboratory, Athens, Georgia  
Instrumental Neutron Activation Analysis

TABLE B-2. Hart Seam Lignite, Major Constituent Analysis Summary.

REMARKS	PROXIMATE ANALYSIS*						ULTIMATE ANALYSIS*				
	MOISTURE %	VOLATILE MATTER %	FIXED CARBON %	ASH %	CARBON %	NITROGEN %	HYDROGEN %	OXYGEN %	SULFUR %	HEATING VALUE (BTU/lb)	
Hart Seam Lignite Analyses from the mine area; 62 analyses (Petrow, 1974).	43.8	35.5	21.9	50.9	0.39	4.1	22.5	0.81	8,600		
Test Burn Lignite from Test Pit #2; composite of 10 samples (CDM/Accu-Labs).	9.88				23.34				0.72	8,385	
Test Burn Lignite; arithmetic mean of 7 individual analyses (U.S. Dept. of Energy Coal Lab.).	9.8	39.0	38.6	22.4	52.4	0.7	3.2	20.8	0.6	8,020	
Test Burn Lignite; duplicate analyses of a single sample (CDM/Accu-Labs).	12.27				22.20				0.76	8,424	
Test Burn Lignite; freshly mined from Test Pit #2 (Boundary Dam Power Station Lab.)	36.55†										
Willow Bunch Lignite Samples; arithmetic mean of 8 analyses (Montana Dept. of Health and Environmental Sciences)	8.0				16.0				0.9	9,138	
Best Values (Average)	36.55†	43.7	35.6	21.4	50.9	0.40	4.1	22.5	0.80	8,596	
Normalized Best Values	36.55	43.4	35.4	21.4	50.8	0.40	4.1	22.5	0.80		

\*All values except moisture content are reported on a moisture free basis.

†This value for the moisture content was selected as the most representative value because it represents a large sample of freshly mined material. All other moisture values shown in the table are from samples subjected to a varying period of air drying.

TABLE B-3. ASH, Major Constituent Analysis Summary.

REMARKS	$\text{SiO}_2$	$\text{Al}_2\text{O}_3$	$\text{TiO}_2$	$\text{Fe}_2\text{O}_3$	$\text{Na}_2\text{O}$	$\text{K}_2\text{O}$	$\text{CaO}$	$\text{MgO}$	$\text{P}_2\text{O}_5$	$\text{SO}_3$
1 Test Burn Ash, samples from 500-ton test burn at Boundary Dam Power Station	52.4	16.6	0.59	4.78	3.20	1.98	14.0	4.43	0.35	1.20
2 Ash from Hart Seam Lignite samples; from throughout mine area	44.4	15.5	1.2	6.8	1.4	1.4	12.4	4.6		
3 Most Representative Values	44.8	15.6	1.2	6.7	1.5	1.4	12.5	4.6	0.35	1.20
4 Values Normalized to 100%	50.0	17.4	1.3	7.5	1.7	1.6	13.9	5.1	0.35	1.20

**Notes:**

1 Analyses of 3 bottom ash, 3 upper ash, and 4 dust collector ash samples multiplied by quantity weighting factors empirically derived from the test burn at the Boundary Dam Power Station.

2 Each of the 62 ash analyses was weighted by the total lignite seam thickness and then averaged.

3 The most representative values were obtained by giving a weight of 3.33 samples to the test burn results and a weight of 62 samples to the Hart Seam analyses.

4 The values of  $\text{P}_2\text{O}_5$  and  $\text{SO}_3$  were held constant and the 8 remaining constituents were proportionately adjusted so that the total of all 10 components is equal to 100 percent.

TABLE B-4. ASH, Minor and Trace Element Analyses

ELEMENT	1 Environmental Protection Agency Corvallis, OR	2 Environmental Protection Agency Athens, GA SSMS	3 Environmental Protection Agency Athens, GA INAA	4 CDM/Accu-Labs Athens, GA CO	Wheat Ridge, CO	Most Representative Value
Antimony (ppm)				2.2	2.0	2.1
Arsenic (ppm)	39		9.7	6.6	18.5	
Barium (%)	0.40		0.46	>0.73	0.53	
Beryllium (ppm)				2.8	2.8	
Boron (ppm)	721	172	0.4	381	425	
Bromine (ppm)	<10	4		1.7	1.9	
Cadmium (ppm)	1	ND		1.8	1.8	
Cerium (ppm)			114		114	
Cesium (ppm)			3.9		3.9	
Chlorine (ppm)		463	463		463	
Chromium (ppm)	27	82	36	94	60	
Cobalt (ppm)		17	5.6	7.1	9.8	
Copper (ppm)	34	36		59	43	
Fluorine (ppm)		41		147	94	
Gallium (ppm)			1,089		1,089	
Gold (ppb)			4.6		4.6	
Hafnium (ppm)			12		12	
Holmium (ppm)			6.2		6.2	
Lead (ppm)		43		18		
Manganese (ppm)	400	219	452	561	408	
Molybdenum (ppm)		2	33		17	
Nickel (ppm)	<50	73		4.4	39	
Rubidium (ppm)			41		41	
Selenium (ppm)	4		2.4	<1.1	<2.6	
Silver (ppm)				0.38	0.38	
Strontium (ppm)	1,825		1,171	>3,163	>2,053	
Tantalum (ppm)			1.7		1.7	
Tellurium (ppm)			94		94	
Thorium (ppm)			14		14	
Tungsten (ppm)			2.5		2.5	
Uranium (ppm)		11	9.5	13	11	
Vanadium (ppm)			74	75	74	
Zinc (ppm)			33	8.7	21	
Zirconium (ppm)	64	217		434	238	

1 U.S. Environmental Protection Agency, Analytical Support Division, Environmental Research Laboratory, Corvallis, Oregon.

2 U.S. Environmental Protection Agency, Environmental Research Laboratory, Athens, Georgia. Spark Source Mass Spectrometry.

3 U.S. Environmental Protection Agency, Environmental Research Laboratory, Athens, Georgia. Instrumental Neutron Activation Analysis.

4 CDM/Accu-Labs, 11485 W. 48th Avenue, Wheat Ridge, Colorado.

## ANNEX C

### WATER ANALYSIS - PUMPWELL WATER QUALITY DATA

Annex C contains 16 tables showing the results of analysis of water from the pumpwells and test pits in the Coronach lignite mining area.



**Table C-1. Water Analyses, SITE 1, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days from First Sample	Ca++	Mg++	Na+	K+	B	Fe	Mn	CO <sub>3</sub> <sup>2-</sup>	SO <sub>4</sub> <sup>2-</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	pH	Laboratory Code	
CH2338	Site #1.....	7/25/75	0	19	84	106	-	-	8.4	<0.01	-	569	205	2	10	-	1,029	392	466	7.9	1
CH2484	...do.....	8/08/75	14	64	66	91	-	-	0.8	0.2	<0.03	578	170	<1	8	-	978	432	474	7.9	1
5466	Site #1 - ditch.....	8/19/75	25	64	63	60	-	1.85	0.2	<0.03	555	120	<0.3	-	-	629	420	455	8.2	2	
5536	Site #1.....	8/22/75	28	56	39	48	-	1.0	0.2	<0.03	423	91	0.9	2	-	461	300	347	7.8	2	
5557	Sample #1, Site 1.....	10/06/75	73	76	53	76	-	1.7	0.2	<0.05	285	130	1.2	20	-	664	410	475	8.0	2	
5576	Sample 10, Site 1.....	10/09/75	76	68	56	83	-	1.4	0.2	<0.03	273	129	1	4	-	643	400	455	8.0	2	
5627	Sample No. 19, Site 1.....	10/24/75	91	69	59	84	10	1.85	0.4	<0.03	280	133	0.9	4	0.26	661	415	466	7.9	2	
5691	Sample No. 31, Site 1.....	11/06/75	104	68	53	83	10	1.6	0.3	<0.05	262	134	<0.3	2	0.25	616	388	435	7.8	2	
5711	Sample No. 41, Site 1.....	11/21/75	119	74	62	87	10	2.4	0.5	0.06	302	140	1.3	2	0.21	674	504	504	7.8	2	
5719	Sample No. 51, Site 1.....	12/05/75	133	76	80	10	2.1	0.3	0.06	n/a	450	439	<0.3	10	0.26	1,470	522	751	7.8	2	
CH1485	S.P.C. Coronach Site 1.....	5/07/76	287	61	59	261	18	-	0.3	<0.01	-	749	360	<1	10	-	1,500	396	614	8.2	1
CH1404	...do.....	5/10/76	290	64	71	257	-	-	0.2	<0.01	-	842	325	<1	12	-	1,571	432	690	8.1	1
5814	...do.....	5/11/76	311	45	77	179	-	1.4	0.5	0.10	n/a	690	251	2.4	6	0.22	980	431	566	8.1	2
5824	...do.....	5/31/76	311	58	60	197	10	-	0.3	0.05	n/a	709	239	2.2	4	0.30	1,014	390	580	8.0	2
5876	...do.....	7/06/76	347	17	11	10	-	0.5	0.18	n/a	107	79	*115	2	<0.1	475	220	88	7.0	2	
5881	...do.....	7/20/76	361	85	54	229	20	2.2	0.2	0.05	n/a	771	297	3.3	8	0.20	1,115	436	631	7.9	2
5913	...do.....	8/02/76	374	102	47	230	4	2.0	1.2	0.08	n/a	792	295	3.5	4	0.27	1,143	447	650	7.5	2
5942	...do.....	8/16/76	388	74	42	222	10	1.8	0.7	0.05	n/a	678	283	<0.3	6	0.24	1,025	359	556	7.7	2
5966	...do.....	9/15/76	418	103	50	213	10	2.2	0.6	0.21	n/a	768	288	<0.3	6	0.31	1,124	461	630	7.5	2
6003	...do.....	10/13/76	446	98	50	222	10	2.1	0.7	0.1	n/a	756	295	3.0	6	0.30	1,135	450	620	7.3	2
6013	...do.....	10/27/76	460	97	54	217	4	2.0	1.4	0.20	n/a	767	301	2.0	4	0.25	1,135	462	628	7.7	2
6027	S.P.C. Coronach Site 1, #127.....	11/10/76	474	98	57	203	10	2.0	0.85	0.31	n/a	751	304	1.6	4	0.27	1,137	478	616	7.7	2
6131	S.P.C. Site 1, Sample No. 188.....	3/23/77	607	98	44	219	10	1.8	4.1	<0.05	n/a	745	290	2.7	4	0.34	1,092	426	611	7.8	2
6145	S.P.C. Coronach Site 1.....	4/14/77	629	61	44	204	10	2.0	0.7	<0.05	n/a	605	283	1.4	4	0.25	1,005	336	495	8.1	2
6186	...do.....	5/12/77	657	102	42	219	10	1.8	1.8	0.05	n/a	773	300	1.9	2	0.30	1,121	426	601	8.0	2
6203	...do.....	5/27/77	672	99	50	223	10	1.7	0.3	<0.03	n/a	747	310	1.9	4	0.30	1,129	454	612	7.8	2
6241	...do.....	6/24/77	700	104	42	216	10	1.7	1.0	0.04	n/a	745	300	2.8	4	0.28	1,141	450	611	8.3	2
6292	...do.....	8/18/77	755	90	52	220	10	1.8	0.45	0.04	n/a	756	290	2.3	4	0.34	1,086	438	620	8.0	2
6335	...do.....	11/09/77	838	93	50	229	4	1.6	0.8	0.22	n/a	765	295	2.4	4	0.30	1,112	438	627	7.9	2
6336	...do.....	11/24/77	853	101	45	214	4	1.7	0.5	0.25	n/a	741	283	2.3	4	0.32	1,100	437	607	7.7	2
6376	...do.....	12/06/77	865	93	56	229	4	1.7	1.6	0.38	n/a	771	311	2.8	4	0.25	1,139	462	632	7.7	2
6395	...do.....	12/21/77	880	94	49	220	6	1.6	0.9	0.30	n/a	750	290	2.8	2	0.30	1,077	437	615	7.9	2
Range.....																					
Arithmetic Mean.....																					
Standard Deviation.....																					
Standard Error of the Mean.....																					

\*Outlying value not included in mean

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan

2. University of Saskatchewan Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-2. Water Analyses, TEST PIT 2, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca ++	Mg ++	K +	B	Fe	Mn	CO <sub>3</sub> -	HCO <sub>3</sub> -	SO <sub>4</sub> -	NO <sub>3</sub> -	F -	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	pH	Laboratory Code	
CW2260	Coronach Test Pit 2 (upper bnd)	7/23/75	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1		
5543	Test Pit 2.....	9/05/75	44	72	39	13	-	0.2	0.08	0.11	362	70	24	5	-	423	340	297	8.1	
5544	...do.....	9/05/75	44	72	39	15	-	0.2	0.15	0.08	362	66	25	5	-	411	340	297	8.0	
5521	Coronach Test Pit, Upper Aquifer	9/12/75	51	76	32	7	-	0.2	0.18	0.11	342	30	55	5	-	-	280	280	280	8.2
5522	Coronach Test Pit, East Sump.....	9/12/75	51	84	27	15	-	0.1	0.1	0.10	310	46	11.5	5	-	386	320	295	7.9	
5562	Test Pit 2, Sample No. 6.....	10/24/75	93	68	34	14	-	0.2	0.53	0.11	172	46	22	20	-	350	310	287	8.0	
5632	Test Pit 2, Sample No. 24.....	10/24/75	93	66	36	14	4	0.1	0.4	0.12	311	51	21	<1	0.53	340	311	278	8.1	
5696	Test Pit 2, Sample No. 36.....	11/06/75	106	90	57	52	8	1.1	0.35	0.37	0.11	271	114	7	5	0.42	609	460	454	7.3
5716	Test Pit 2, Sample No. 48.....	11/21/75	121	97	51	80	10	1.8	<0.1	0.28	0.11	287	147	1.6	2	0.21	702	454	479	7.3
5717	Test Pit 2, Sample No. 49.....	12/05/75	135	92	54	79	10	1.6	0.32	0.11	297	135	0.6	5	0.22	706	452	495	7.3	
5733	Test Pit 2, Sample No. 58.....	12/17/75	147	92	56	83	10	1.5	0.35	0.11	301	143	5	<1	0.31	651	460	502	7.2	
5734	Test Pit 2, Sample No. 59.....	12/17/75	147	111	93	137	10	1.5	0.22	0.11	380	315	3.1	2	0.26	1,117	660	635	7.4	
5819	Test Pit 2.....	5/20/76	302	58	40	31	4	0.9	0.6	0.10	361	78	4.2	2	0.27	432	308	296	7.0	
5835	...do.....	5/31/76	313	79	50	56	4	0.9	0.45	0.20	508	103	2.2	2	0.28	641	404	416	7.5	
Range.....																				
		58-	27-	7-	4-	0.1-	<0.1-	0.10-	n11	168-	315	70	20	0.6-	<1-	0.21-	340-	310-	278-	
		111.	93	137	10	1.8	4.4	0.53	508	508	53	1,117	660	53	1,117	660	592	592	8.2	
		81	47	46	8	0.8	1.1	0.23	-	318	103	18	<5	0.31	564	395	385	7.6		
		15	17	40	3	0.7	1.5	0.14	-	92	75	21	5	0.11	224	102	118	0.4		
		44	±5	±11	±1	±0.2	±0.4	±0.04	-	±27	±21	±5.6	±1	±0.04	±65	±28	±33	±0.1		

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan
2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-3. Water Analyses, KESSLER CROSSING SITE 3, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days from First Sample	Ca <sup>++</sup>	Mg <sup>++</sup>	Na <sup>+</sup>	K <sup>+</sup>	B	Fe	Mn	CO <sub>3</sub> <sup>2-</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>2-</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	Laboratory Code
6352	Sample No. 391, Kessler Crossing	11/24/77	0	99	52	230	4	2.0	0.2	0.31	nil	782	311	2.4	2	0.30	1,158	464	8.1	2
6378	Sample No. 404, Kessler Crossing	12/06/77	12	107	47	231	4	1.8	0.1	0.31	nil	786	308	2.9	2	0.28	1,179	461	64.4	
6394	Sample No. 412, Kessler X-ing...	12/21/77	27	102	47	231	7	1.4	0.25	0.15	nil	762	313	3.0	4	0.30	1,139	44.9	62.5	2
6497	Sample No. 491, Kessler X-ing...	3/20/78	116	27	7	13	10	0.1	0.6	0.08	nil	116	35	4.1	1	<0.1	197	97	7.9	2
6506	Sample No. 510, Kessler Crossing	4/04/78	131	49	25	114	8	0.6	0.5	0.09	nil	340	193	2.0	2	0.13	620	225	279	2
Range.....																				
		107	52	231	10	2.0	0.2	0.1-	0.1-	0.31	nil	786	313	2.9	2	<0.1-	1,179	464	64.4	
Arithmetic Mean.....																				
		77	36	164	7	1.2	0.3	0.19	0.19	nil	557	232	2.9	2	<0.22	859	339	457	7.8	
Standard Deviation.....																				
		36	19	98	3	0.8	0.2	0.11	0.11	-	311	121	0.8	1	0.10	437	169	255	0.3	
Standard Error of the Mean.....																				
		±16	±9	±64	±1	±0.4	±0.1	±0.05	±0.05	-	±139	±56	±0.4	±1	±0.04	±196	±76	±114	±0.2	

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan
2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-4. Water Analyses, DUNCAN CROSSING SITE 5, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca <sup>++</sup>	Mg <sup>++</sup>	Na <sup>+</sup>	K <sup>+</sup>	B	P	Mn	CO <sub>3</sub> <sup>++</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	Laboratory Code		
6355	Sample No. 394, Duncan Crossing	11/24/77	0	119	66	269	4	2.2	0.4	0.33	n.d.	927	375	3.5	4	0.35	1,374	568	760	7.8	
6377	Sample No. 403, Duncan X'ing...	12/06/77	12	123	45	242	4	1.8	0.3	0.26	n.d.	831	321	3.0	4	0.29	1,219	492	681	7.7	
6393	Sample No. 441, Duncan X'ing...	12/21/77	27	111	47	221	6	1.6	0.35	0.26	n.d.	775	303	2.8	4	0.28	1,128	470	635	7.8	
6496	Sample No. 496, Duncan X'ing...	3/20/78	116	24	9	15	10	0.1	0.6	0.08	n.d.	121	38	4.2	1	0.46	214	98	99	7.3	
6507	Sample No. 511, Duncan Crossing	4/04/78	131	45	30	109	8	0.6	0.6	0.09	n.d.	344	191	2.2	2	0.17	627	237	282	7.7	
Range.....																			99-	99-	
Arithmetic Mean.....																			760	568	7.8
Standard Deviation.....																			373	491	7.7
Standard Error of the Mean.....																			286	0.2	

**Table C-5. Water Analyses, PUMPWELL 2-CULLVERT, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca <sup>++</sup>	Mg <sup>++</sup>	Na <sup>+</sup>	K <sup>+</sup>	B	Fe	Mn	CO <sub>3</sub> <sup>=</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	pH	Laboratory Code
6356	Sample No. 395, PW2-Culvert..	1/12/77	0	101	45	214	4	1.7	0.5	0.25	nil	741	283	2.3	4	0.32	1,100	437	607	7.9	2
6376	Sample No. 402, P.W. 2 Culvert	1/2/06/77	12	92	56	229	4	1.7	1.6	0.38	nil	771	311	2.8	4	0.25	1,139	462	632	7.7	2
6395	Sample No. 443, P.W. 2 Culvert	1/2/21/77	27	94	49	220	6	1.6	0.9	0.30	nil	750	290	2.8	2	0.30	1,077	437	615	7.9	2
6423	Sample No. 451, P.W. 2 Culvert			90	47	220	6	1.8	0.8	0.24	nil	723	290	3.2	4	0.32	1,055	417	593	8.2	2
6445	Sample No. 473, P.W. 2 Culvert	2/17/78	85	94	52	204	6	0.75	0.75	0.31	nil	734	282	2.6	2	0.32	1,101	449	602	7.9	2
6475	Sample No. 486, P.W. 2 Culvert			98	45	221	6	1.7	0.6	0.14	nil	753	282	2.5	4	0.32	1,068	430	617	7.6	2
6480	Sample No. 492, P.W. 2 Culvert			94	50	221	6	1.7	1.0	0.22	nil	764	265	3.0	2	0.24	1,072	440	626	7.8	2
6508	Sample No. 512, P.W. 2 Culvert	4/04/78	131	94	50	221	6	1.3	0.8	0.09	nil	745	299	3.0	4	0.33	1,107	441	611	8.0	2
Range.....																					
				90-	45-	204-	4-	1.3-	0.5-	0.09-	-	723-	282-	2.3-	2-	0.25-	1,055-	417-	593-	7.6-	
				101	56	229	6	2.0	1.6	0.38	-	771	311	3.2	4	0.34	1,139	462	632	8.2	
Arithmetic Mean.....																					
				95	49	219	6	1.7	0.9	0.24	nil	748	290	2.8	3	0.31	1,090	439	613	7.9	
Standard Deviation.....																					
				3	4	7	1	0.2	0.3	0.09	-	16	10	0.3	1	0.03	27	13	13	0.2	
Standard Error of the Mean.....																					
				+1	+1	+3	0	0.0	+0.1	+0.03	-	+5	+4	+0.1	0	+0.01	+10	+5	+4	+0.1	

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan
2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-6. Water Analyses, PUMPWELL 1, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Na <sup>+</sup>	K <sup>+</sup>	B	Fe	Mn	CO <sub>3</sub> <sup>=</sup>	HC O <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	PH	Laboratory Code		
6345	Sample No. 384, PW-1..	11/24/77	0	109	52	195	4	2.0	1.2	0.40	n11	765	273	1.5	2	0.30	1,068	488	627	7.6	2
6372	Sample No. 396, P.W.-1	12/06/77	12	103	54	202	4	2.4	1.2	0.34	n11	772	273	4.9	2	0.24	1,061	479	633	7.3	2
6390	Sample No. 438, P.W.-1	12/21/77	27	109	52	195	6	1.7	1.3	0.39	n11	770	268	6.1	2	0.29	1,069	486	631	7.5	2
6416	Sample No. 444, P.W.-1			101	61	194	6	2.2	1.2	0.36	n11	767	284	3.6	4	0.31	1,089	504	629	7.8	2
6454	Sample No. 472, P.W.-1	2/13/78	81	96	61	193	6	2.0	1.75	0.37	n11	756	284	3.8	2	0.31	1,092	493	620	7.5	2
6473	Sample No. 481, P.W.-1			103	52	197	6	1.8	1.6	0.14	n11	742	278	2.9	2	0.26	1,038	471	611	7.5	2
6479	Sample No. 491, P.W.-1			102	55	204	6	2.2	1.4	0.35	n11	766	284	4.2	4	0.33	1,069	480	628	7.7	2
Range.....																					
				96-	52-	193-	4-	1.7-	1.2-	0.14-	-	742-	268-	1.5-	2-	0.24-	1,038-	471-	611-		
				109	61	204	6	2.4	1.75	0.40	-	772	284	6.1	4	0.33	1,092	504	633	7.8	
Arithmetic Mean.....																					
				103	55	197	5	2.0	1.4	0.34	n11	763	278	3.9	3	0.29	1,072	486	626	7.6	
Standard Deviation.....																					
				5	4	4	1	0.2	0.2	0.09	-	10	7	1.5	2	0.03	13	11	8	0.2	
Standard Error of the Mean.....																					
				+2	+2	+2	0	+0.1	+0.1	+0.03	-	+4	+2	+0.6	+1	+0.01	+5	+4	+3	+0.1	

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan

2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-7. Water Analyses, PUMPWELL 2, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca <sup>++</sup>	Mg <sup>++</sup>	K <sup>+</sup>	B	Fe	Mn	CO <sub>3</sub> <sup>=</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	pH	Laboratory Code	
6349	Sample No. 388, P.W. 2	11/24/77	0	59	242	4	1.4	0.30	n11	797	366	4.1	2	0.27	1,264	509	653	7.4	2	
6374	Sample No. 400, P.W. 2	12/06/77	12	107	58	250	5	1.7	0.34	n11	806	370	4.2	4	0.24	1,255	506	661	7.4	
6388	Sample No. 436, P.W. 2	12/21/77	27	119	52	242	6	1.3	0.31	n11	797	370	2.6	4	0.30	1,251	512	653	7.6	2
6417	Sample No. 445, P.W. 2			119	52	241	6	1.6	0.28	n11	782	374	3.2	6	0.29	1,253	510	641	7.8	2
6459	Sample No. 477, P.W. 2	2/17/78	85	106	57	241	6	1.6	0.30	n11	775	366	2.1	10	0.30	1,250	500	635	7.3	2
6472	Sample No. 483, P.W. 2			106	64	232	6	1.6	0.28	n11	778	379	2.6	4	0.29	1,221	530	638	7.7	2
6502	Sample No. 506, P.W. 2	4/04/78	131	103	53	223	6	1.3	0.26	n11	692	377	1.8	4	0.30	1,227	474	567	7.4	2
Range.....																				
				103-	52-	223-	4-	1.3-	1.1-	0.26-	-	692-	366-	1.8-	2-	0.24-	1,221-	474-	567-	7.3-
				119	64	250	6	1.7	3.5	0.34	-	806	379	4.2	10	0.30	1,264	530	661	7.7
Arithmetic Mean.....																				
				110	56	239	6	1.5	1.8	0.30	n11	775	372	2.9	5	0.28	1,246	506	635	7.5
Standard Deviation.....																				
				7	4	9	1	0.2	0.8	0.03	-	38	5	0.9	3	0.02	16	17	32	0.2
Standard Error of the Mean.....																				
				+2	+2	+3	0	+0.1	+0.3	+0.01	-	+15	+2	+0.4	+1	+0.01	+6	+6	+12	+0.1

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan

2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-8. Water Analyses, PUMPWELL 6, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca <sup>++</sup>	Mg <sup>++</sup>	Na <sup>+</sup>	K <sup>+</sup>	B	Fe	Mn	CO <sub>3</sub> <sup>=</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	Laboratory Code	
6347	Sample No. 386, P.W. 6	11/24/77	0	1.24	69	189	4	2.3	1.0	0.35	n.i.l	776	356	1.2	2	0.26	1,193	594	636	7.3	2
6379	Sample No. 405, P.W. 6	12/06/77	12	1.37	66	202	4	2.0	1.0	0.39	n.i.l	831	358	<0.2	4	0.23	1,269	616	681	7.1	2
6392	Sample No. 440, P.W. 6	12/21/77	27	1.34	66	202	7	2.3	4.8	0.36	n.i.l	824	361	3.2	2	0.25	1,238	607	675	7.5	2
6418	Sample No. 446, P.W. 6			1.31	72	196	6	2.2	1.5	0.34	n.i.l	828	360	<0.2	2	0.27	1,250	622	679	7.6	2
6456	Sample No. 474, P.W. 6	2/17/78	85	1.34	66	186	6	2.2	7.5	0.36	n.i.l	792	350	5.4	2	0.26	1,256	608	649	7.1	2
6470	Sample No. 481, P.W. 6			1.42	64	194	7	2.7	1.0	0.16	n.i.l	738	418	3.2	2	0.25	1,237	616	605	7.8	2
6477	Sample No. 489, P.W. 6			1.19	72	188	6	2.6	0.9	0.15	n.i.l	741	383	1.2	2	0.26	1,193	594	607	7.6	2
5504	Sample No. 508, P.W. 6	4/04/78	131	1.24	67	189	6	2.3	2.5	0.40	n.i.l	749	369	3.2	4	0.30	1,209	586	614	7.3	2
Range.....																					
				119-	66-	186-	4-	2.0-	0.9-	0.15-	-	738-	350-	<0.2-	2-	0.23-	1,193-	586-	605-	7.1-	
				142	72	202	7	2.7	7.5	0.40		831	418	5.4	4	0.27	1,269	616	681	7.8	
Arithmetic Mean.....																					
				131	68	193	6	2.3	2.5	0.31	n.i.l	785	369	<2.2	3	0.26	1,227	605	643	7.4	
Standard Deviation.....																					
				8	3	6	1	0.2	2.4	0.10	-	40	22	1.8	1	0.02	27	13	33	0.3	
Standard Error of the Mean.....																					
				+3	+1	+2	0	+0.1	+0.9	+0.04	-	+14	+8	+0.6	0	+0.01	+10	+5	+12	+0.1	

**Table C-9. Water Analyses, PUMPWELL 8, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Chemical Analysis										Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	pH	Laboratory Code				
				Ca <sup>++</sup>	Mg <sup>++</sup>	Na <sup>+</sup>	K <sup>+</sup>	B	Fe	Mn	CO <sub>3</sub> <sup>2-</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>2-</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>					
6346	Sample No. 385, PW 8	11/24/77	0	92	48	269	4	2.2	0.9	0.40	n/a	826	325	2.6	2	0.35	1,250	429	677	7.6	2
6373	Sample No. 399, P.W. 8	12/06/77	12	89	48	282	4	2.0	1.1	0.38	n/a	839	329	3.0	4	0.30	1,217	421	688	7.4	2
6389	Sample No. 431, P.W. 8	12/21/77	27	94	45	272	6	1.7	3.0	0.40	n/a	815	328	3.2	4	0.35	1,226	419	668	7.4	2
6419	Sample No. 447, P.W. 8	103	45	272	6	2.3	1.1	0.40	n/a	815	349	3.5	4	0.35	1,262	440	668	7.9	2		
6501	Sample No. 505, P.W. 8	4/04/78	131	123	30	262	6	1.8	1.0	0.35	n/a	800	334	4.2	2	0.36	1,210	430	656	7.5	2
Range.....				89-	30-	262-	4-	1.7-	0.9-	0.35-	-	800-	325-	2.6-	2-	0.30-	1,210-	419-	656-	7.4-	
Arithmetic Mean.....				123	48	282	6	2.3	3.0	0.40		839	349	3.5	4	0.36	1,262	440	688	7.9	
Standard Deviation.....				100	43	271	5	2.0	1.4	0.39	n/a	819	333	3.3	3	0.34	1,233	428	671	7.6	
Standard Error of the Mean.....				14	8	7	1	0.3	0.9	0.02	-	15	10	0.6	1	0.02	22	8	12	0.2	
				+6	+3	+3	+1	+0.1	+0.4	+0.01	-	+6	+4	+0.3	+1	+0.01	+10	+4	+5	+0.1	

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan
2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-10. Water Analyses, PUMPWELL 10A, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca++	Mg++	Na+	K+	8	Fe	Mn	CO <sub>3</sub> <sup>=</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	pH	Laboratory Code	
6350	Sample No. 389, PW 10A	11/24/77	0	96	49	213	4	2.0	1.5	0.40	n.d.	739	288	1.1	2	0.30	1,086	442	606	7.5	
6370	Sample No. 396, P.W. 10A	12/06/77	12	93	51	221	4	2.0	1.7	0.40	n.d.	747	291	4.4	6	0.25	1,079	441	612	7.3	
6386	Sample No. 438, P.W. 10A	12/21/77	27	102	47	212	6	1.8	1.2	0.40	n.d.	736	294	3.2	4	0.30	1,079	449	603	7.4	
6420	Sample No. 449, P.M. 10A			103	45	213	6	1.6	1.3	0.38	n.d.	732	293	4.2	4	0.31	1,083	441	600	7.8	
6457	Sample No. 475, P.M. 10A	2/17/78	85	90	86	47	220	6	1.7	1.4	0.34	n.d.	693	291	2.6	2	0.34	1,085	408	568	7.4
6474	Sample No. 485, P.W. 10A			86	52	213	6	1.8	2.5	0.36	n.d.	706	295	2.8	4	0.33	1,081	408	579	7.9	
6478	Sample No. 490, P.W. 10A			86	52	213	6	1.6	0.9	0.32	n.d.	717	293	1.9	4	0.34	1,057	430	588	7.1	
6503	Sample No. 507, P.W. 10A	4/04/78	131	86	52	213	6	1.6	0.9	0.32	n.d.	715	296	3.0	4	0.26	1,076	430	586	7.5	
Range.....																					
Arithmetic Mean.....																				7.1-7.9*	
Standard Deviation.....																					
Standard Error of the Mean.....																					

**Laboratory Code**

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan

2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-11. Water Analyses, PUMPWELL 11, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca <sup>++</sup>	Hg <sup>++</sup>	Na <sup>+</sup>	K <sup>+</sup>	B	Fe	Mn	C0 <sub>3</sub> <sup>=</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	Total pH	Laboratory Code
6354	Sample No. 393, PW 11	11/24/77	0	58	187	4	1.3	0.36	n11	772	282	<0.2	4	0.27	1,118	521	633	7.2	2	
6380	Sample No. 406, P.W. 11	12/06/77	12	118	54	194	4	2.0	1.0	788	284	<0.2	2	0.24	1,121	516	646	7.2	2	
6625	Sample No. 453, P.W. 11	113	58	184	6	1.8	1.5	0.32	n11	760	288	<0.2	4	0.34	1,090	521	623	7.8	2	
6499	Sample No. 503, P.W. 11	4/04/78	131	58	179	6	1.7	1.1	0.33	n11	759	275	0.4	4	0.30	1,087	516	622	7.4	2
Range.....																				
		112-	54-	179-	4-	1.7-	1.0-	0.32-	-	759-	275-	<0.2-	2-	0.24-	1,087-	516-	622-	7.2-		
		118	58	194	6	2.0	1.5	0.38	-	788	288	0.4	4	0.30	1,121	521	646	7.8		
Arithmetic Mean.....																				
		114	57	186	5	1.8	1.2	0.35	n11	770	282	0.3	4	0.34	1,104	519	631	7.4		
Standard Deviation.....																				
		3	2	6	1	0.1	0.2	0.03	-	14	5	0.1	1	0.12	18	3	11	0.3		
Standard Error of the Mean.....																				
		±1	±1	±3	±1	±0.1	±0.1	±0.01	-	±7	±3	±0.1	±1	±0.06	±9	±1	±6	±0.1		

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan
2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-12. Water Analyses, PUMPWELL 12A, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca <sup>++</sup>	Mg <sup>++</sup>	Na <sup>+</sup>	K <sup>+</sup>	B	Fe	Mn	CO <sub>3</sub> <sup>=</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	pH	Laboratory Code
6353	Sample No. 392, PW 12A	11/24/77	0	113	48	250	4	2.0	1.4	0.40	n.d.	803	351	2.2	4	0.32	1,233	481	638	7.6	2
6381	Sample No. 407, P.W. 12A	12/06/77	12	120	48	244	4	2.0	1.0	0.40	n.d.	813	346	<0.2	4	0.29	1,250	496	666	7.2	2
6426	Sample No. 452, P.W. 12A	114	47	249	6	2.0	1.5	0.37	n.d.	787	359	<0.2	4	0.32	1,234	476	645	7.9	2		
6481	Sample No. 491, P.W. 12A	102	55	251	6	1.8	1.5	0.33	n.d.	802	354	1.4	6	0.36	1,230	480	657	7.5	2		
6498	Sample No. 502, P.W. 12A	4/04/78	131	110	54	249	6	1.6	1.3	0.36	n.d.	809	356	5.7	6	0.33	1,232	497	663	7.5	2
Range.....																					
Arithmetic Mean.....																					
Standard Deviation.....																					
Standard Error of the Mean.....																					

## Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan  
 2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-13. Water Analyses, PUMPWELL 13, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca <sup>++</sup>	Mg <sup>++</sup>	Na <sup>+</sup>	K <sup>+</sup>	B	Fe	Mn	CO <sub>3</sub> <sup>=</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (aa CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	Total Hardness (aa CaCO <sub>3</sub> )	Total pH	Laboratory Code
6351	Sample No. 390, PW 13	11/24/77	0	90	48	230	4	1.7	1.4	0.38	n.d.	737	306	1.2	4	0.32	1,014	423	604	7.4	2
6371	Sample No. 391, P.W. 13	12/06/77	12	94	45	232	4	2.7	1.9	0.34	n.d.	737	305	2.4	4	0.26	1,106	419	604	7.4	2
6387	Sample No. 435, P.W. 13	12/21/77	27	94	44	230	6	1.3	1.2	0.34	n.d.	730	306	2.8	4	0.28	1,098	418	599	7.7	2
6421	Sample No. 449, P.W. 13	90	47	230	6	1.6	1.0	0.34	n.d.	726	307	2.3	6	0.31	1,108	417	596	7.8	2		
6505	Sample No. 509, P.W. 13	4/04/78	131	89	52	219	6	1.3	0.25	n.d.	723	307	2.6	4	0.30	1,107	436	593	7.5	2	
Range.....	.....	.....	.....	89-	44-	219-	4-	1.3-	1.0-	0.25-	-	723-	305-	1.2-	4-	0.26-	1,014-	417-	593-	7.4-	2
Arithmetic Mean.....	.....	.....	.....	94	52	230	6	2.7	1.9	0.38	737	307	2.8	6	0.32	1,108	436	604	7.4	2	
Standard Deviation.....	.....	.....	.....	91	47	228	5	1.7	1.4	0.33	n.d.	731	306	2.3	4	0.29	1,087	423	599	7.6	2
Standard Error of the Mean.....	.....	.....	.....	2	3	5	1	0.6	0.3	0.05	-	6	1	0.6	1	0.02	41	8	5	0.2	2
				±1	±1	±2	±1	±0.3	±0.2	±0.02	-	±3	0	±0.3	0	±0.01	±18	±2	±2	±0.1	2

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan  
2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-14. Water Analyses, PUMPWELL 14, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca <sup>++</sup>	Mg <sup>++</sup>	Na <sup>+</sup>	K <sup>+</sup>	B	Fe	Mn	CO <sub>3</sub> <sup>=</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Laboratory Code		
6348	Sample No. 387, PW 14	11/24/77	0	94	4.1	254	4	2.0	1.9	0.43	n.d.	811	276	5.4	2	0.34	1,135	403	665	7.5	
6375	Sample No. 401, PW 14	12/06/77	12	86	48	260	4	2.0	1.6	0.48	n.d.	832	283	5.1	4	0.32	1,156	415	682	7.4	
6391	Sample No. 439, PW 14	12/21/77	27	88	39	256	6	1.7	1.5	0.43	n.d.	780	288	3.0	2	0.34	1,146	380	639	7.6	
6422	Sample No. 450, PW 14	1/17/78	37	94	262	6	2.0	1.5	0.40	n.d.	814	281	2.7	4	0.36	1,155	389	667	7.9		
6458	Sample No. 476, PW 14	2/17/78	85	44	248	6	1.8	1.6	0.40	n.d.	784	288	2.8	4	0.28	1,172	408	643	7.4		
6471	Sample No. 482, PW 14	85	49	248	6	1.7	1.6	0.40	n.d.	838	254	3.1	4	0.35	1,152	413	687	7.4			
6476	Sample No. 488, PW 14	93	42	249	6	1.8	1.8	0.42	n.d.	841	251	0.8	2	0.37	1,140	405	689	8.1			
6510	Sample No. 519, PW 14	4/06/78	133	78	50	263	6	2.0	1.7	0.38	n.d.	827	286	<0.3	4	0.34	1,147	401	678	7.7	
Range.....		78-	37-	248-	4-	1.7-	1.5-	0.38-	-	780-	251-	<0.3-	2-	0.28-	5.4	4	0.37	1,135-	380-	639-	7.4-
Arithmetic Mean.....		94	50	263	6	2.0	1.9	0.48		841	288	2.9	3	0.34	1,150	402	689	8.1			
Standard Deviation.....		88	44	255	6	1.9	1.7	0.42	n.d.	816	276	2.9	3	0.34	1,172	415	669	7.6			
Standard Error of the Mean.....		5	5	6	1	0.1	0.1	0.03	-	23	15	1.8	1	0.03	11	12	19	0.3			

## Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan  
 2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

**Table C-15. Water Analyses, PUMPWELL 212, Poplar River Power Project**

Sample No.	Remarks	Sample Date	Days From First Sample	Ca++	Mg++	Na+	K+	B	Fe	Mn	CO <sub>3</sub> =	SO <sub>4</sub> =	NO <sub>3</sub> -	Cl-	F-	Dissolved Solids (as CaCO <sub>3</sub> )	Total Hardness (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	Total pH	Laboratory Code	
CW2025	Pumpwell 212.....	7/08/75	0	51	49	29	-	-	0.3	0.08	-	425	43	3	6	-	606	328	7.7	1	
CW2043	Coronach Test Pit 2 (Well 212)....	7/10/75	2	38	83	69	10	-	0.5	0.27	-	473	190	2	10	-	876	388	7.1	1	
CW2259	Well 212.....	7/23/75	15	77	61	78	-	-	0.9	0.27	-	610	124	1	8	-	960	500	7.5	1	
CW2339	...do.....	7/25/75	17	16	98	79	-	-	1.5	0.18	-	598	125	<1	10	-	960	444	8.2	1	
CW2341	...do.....	7/28/75	20	18	97	79	-	-	1.4	0.23	-	600	120	1	10	-	939	444	6.9	1	
CW2343	...do.....	7/30/75	22	16	97	68	-	-	0.9	0.22	-	586	96	3	6	-	884	440	7.0	1	
CW2480	...do.....	8/01/75	24	80	59	82	-	-	1.0	0.20	-	150	270	<1	10	-	968	444	4.80	1	
CW2482	...do.....	8/03/75	26	83	66	81	-	-	0.5	0.19	-	488	6	6	6	-	995	4.80	4.00	1	
5334	212.....	9/03/75	59	56	78	67	-	-	1.5	1.2	0.08	n.d.	610	126	0.4	10	-	860	500	7.0	2
5519	Coronach Test Pit N.W. Well (212)	9/12/75	66	84	56	60	-	-	1.6	0.45	0.25	n.d.	603	122	2.0	5	-	684	440	4.95	2
5335	212.....	9/22/75	76	48	78	67	-	-	1.7	0.35	0.16	n.d.	598	122	0.7	5	-	585	440	7.0	2
5364	212 Pumpwell, Sample No. 8.....	10/01/75	85	40	68	67	-	-	1.5	0.9	0.21	n.d.	294	127	1.2	2	-	659	380	7.0	2
5381	Pumpwell 212, Sample No. 15.....	10/09/75	93	100	46	75	-	-	1.6	0.6	0.10	n.d.	294	130	1	2	-	692	440	4.90	2
5333	Pumpwell 212, Sample No. 25.....	10/24/75	108	93	51	75	10	1.85	1.2	0.22	n.d.	294	130	0.8	<1.0	0.23	710	444	4.89	2	
5697	Pumpwell 212, Sample No. 31.....	11/06/75	121	88	71	10	1.6	0.8	0.24	n.d.	285	131	0.9	5	0.26	689	448	4.75	2		
5714	Pumpwell 212, Sample No. 46.....	11/21/75	136	82	59	66	10	1.8	0.85	0.24	n.d.	282	127	1.7	<1.0	0.19	669	450	4.69	2	
Range.....																					
Arithmetic Mean.....																					
Standard Deviation.....																					
Standard Error of the Mean.....																					

Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan

2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

Table C-16. Water Analyses, PUMPWELL 213, Poplar River Power Project

Sample No.	Remarks	Sample Date	Days From First Sample	Ca++	Mg++	K+	B	Fe	Mn	CO <sub>3</sub> <sup>=</sup>	HCO <sub>3</sub> <sup>-</sup>	SO <sub>4</sub> <sup>=</sup>	NO <sub>3</sub> <sup>-</sup>	Cl <sup>-</sup>	F <sup>-</sup>	Total Dissolved Solids (as CaCO <sub>3</sub> )	Total Alkalinity (as CaCO <sub>3</sub> )	Total pH	Laboratory Code		
CH2261	Well 213,.....	7/23/75	0	80	63	96	-	-	1.10	0.23	-	654	155	3	8	-	1,060	460	536	7.5	
CH2340	...do.....	7/25/75	2	16	98	95	-	-	1.00	0.24	-	615	125	<1	12	-	985	446	504	7.0	
CH2342	...do.....	7/28/75	5	16	99	95	-	-	1.00	0.23	-	669	128	<1	8	-	1,129	448	548	7.0	
CH2344	...do.....	7/30/75	7	16	98	96	-	-	1.20	0.17	-	646	130	1	4	-	1,005	444	530	7.0	
CH2481	...do.....	8/01/75	9	80	64	97	-	-	3.80	0.14	-	649	128	5.00	8	-	1,035	464	532	7.3	
CH2483	...do.....	8/03/75	11	86	57	96	-	-	0.90	0.20	-	654	124	<1.00	8	-	1,026	452	536	7.1	
5542	213,.....	8/22/75	30	80	61	79	-	-	1.5	0.20	0.10	n/a	641	133	3.20	5	-	721	525	525	7.1
5541	213,.....	9/05/75	44	88	58	83	-	-	1.6	0.25	0.12	n/a	647	131	0.90	5	-	734	460	530	7.0
5520	Coronach Test Pit N.E. Well (213)	9/12/75	51	88	58	71	-	-	1.7	0.45	0.22	n/a	647	131	0.90	5	-	734	460	530	7.0
5563	213 Pump well, Sample No. 7.....	9/6/75	51	96	51	79	-	-	1.7	0.30	0.19	n/a	315	134	1.00	1	-	698	450	525	7.0
5582	Pumpwell 212, Sample No. 16.....	10/09/75	78	92	56	92	-	-	1.7	0.50	0.19	n/a	304	136	0.90	6	-	738	460	515	7.1
5634	Pumpwell 212, Sample No. 26.....	10/24/75	93	90	57	90	12	2.0	1.00	0.20	0.11	318	138	0.80	<1	0.28	759	462	531	7.0	
5698	Pumpwell 212, Sample No. 38.....	11/06/75	106	95	53	91	10	1.6	0.45	0.22	0.11	313	139	1.70	<1	0.34	747	455	522	6.9	
5715	Pumpwell 212, Sample No. 47.....	11/21/75	121	99	86	86	10	1.8	1.00	0.23	0.11	322	136	1.10	<1.0	0.22	670	475	538	6.9	
5718	Pumpwell 213, Sample No. 50.....	12/05/75	135	99	63	91	10	1.6	0.40	0.23	0.11	330	159	<0.30	4	0.23	795	506	551	6.9	
Range.....	.....	.....	16-	53-	71-	10-	1.5-	0.2-	0.1-	-	-	309-	124-	<0.3-	<1-	0.22-	670-	444-	504-	6.9-	
Arithmetic Mean.....	.....	.....	99	99	97	12	2.0	3.80	0.24	669	159	5.00	12	0.34	1,060	506	551	7.5			
Standard Deviation.....	.....	.....	75	66	89	11	1.7	0.90	0.19	515	136	<1.76	<5	0.27	848	459	530	7.1			
Standard Error of the Mean.....	.....	.....	31	17	7.9	1	0.2	0.87	0.04	-	167	10	1.45	3	0.06	151	15	12	0.2		
			+8	+4	+2	+1	+0.5	+0.23	+0.01	-	+43	+2	+0.37	+9	+0.03	+39	+4	+3	0.0		

## Laboratory Code

1. Saskatchewan Department of Public Health Laboratory, Regina, Saskatchewan

2. Sanitary Engineering Laboratory, University of Saskatchewan, Saskatoon, Saskatchewan

## ANNEX D

### MINE SPOIL AND RESERVOIR SOIL LEACHING TEST RESULTS

Annex D contains the following:

- (1) A report entitled "Characterization of Poplar River Mine Spoil and Reservoir Soil Samples" by Dr. G.K. Pagenkopf, Montana State University, May 17, 1978.

(This report has been re-typed to reduce space. Material on coal ash leaching contained in the original report has been removed and is now included only in Dr. Pagenkopf's report in Annex E).
- (2) A report entitled "Cookson Reservoir Leaching Study" summarizing the results of a soil leaching study conducted for the Saskatchewan Power Corporation by Sheppard T. Powell Consultants Ltd. in 1975.



# CHARACTERIZATION OF POPLAR RIVER MINE SPOIL AND RESERVOIR SOIL SAMPLES

Dr. Gordon K. Pagenkopf  
Department of Chemistry  
Montana State University  
Bozeman, Montana 59715  
May 17, 1978

The data included in this report summarize the results of a series of leaching studies conducted on Poplar River mine spoil and reservoir soils. The work is under contract with Environment Canada at the request of the International Poplar River Water Quality Board Plant, Mine, and Reservoir Operations Committee. Mr. Dan B. Kimball, U.S. Environmental Protection Agency, Denver and Mr. Wayne Draper, EPS-Ottawa, have been my major contacts.

## Scope of Experiments

A total of fifty-one spoil and soil samples have been leached using column and static techniques. The leachates have been analyzed for major inorganic constituents, nine trace metals and organic carbon.

## Soil Samples

In a previous study we received portions of core samples from the Saskatchewan Power Corporation. These were composited and subsequently leached and the leachate analyzed for major components. In many cases, a minimal quantity of a particular core segment was available and thus this necessitated slightly different compositing. The composites are listed in Table I. The sample number followed by an A, for example 12A, was prepared similar to 12 except none of the 60-70 cm cut was available. In every case, equal weights of soils were mixed using the "rolling" technique. An alternate sample numbering summary is presented in Table II.

Fifteen additional soil samples were obtained from Saskmont Engineering, Regina. These are designated PR 1 through PR 15. Additional details regarding location and collection procedures are presumably available from Saskmont.

## Major Component Leaching

In the previous study samples 1-36 were leached using column and static techniques. That data is included in the report, Table III and Table IV, for comparison purposes. The flow rate for the column leaching studies ranged from 50 to 100 ml per hour, depending on soil type. The samples were leached until the specific conductance of the effluent had decreased to 10% of the initial value. Comparable amounts of water and soil were utilized for the static leachings.

TABLE I COMPOSITE SOIL SAMPLES FROM POPLAR RIVER

<u>Sample Number</u>	<u>Alternate Number</u>	<u>Sample Number</u>	<u>Alternate Number</u>
1	37	22	
2	38	22A	58
3	39	23	59
4	40	24	
5	41	24A	60
6	42	25	61
7	43	26	62
8	44	27	63
9	45	28	
10	46	28A	64
11	47	29	65
12		30	66
12A	48	31	67
13	49	32	68
14		33	69
14A	50	34	
15	51	34A	70
16	52	35	71
17		36	72
17A	53		
18	54		
19	55		
20	56		
21			
22	57		

TABLE II SAMPLE NUMBERING CROSS REFERENCE

<u>Composite No.</u>	<u>Hole No.</u>	<u>Sample No.</u>
1	T.H. 101	F-72, F-73, F-74
2	T.H. 101	F-75, F-76, F-77
3	T.H. 101	F-78, F-79, F-80, F-81, F-82
4	T.H. 300	F-1, F-2, F-3
5	T.H. 300	F-4, F-5, F-6
6	T.H. 300	F-7, F-8, F-9
7	T.H. 201	F-132, F-133, F-134
8	T.H. 201	F-135, F-136, F-137, F-138 F-139, F-140, F-141, F-143 F-144, F-145, F-146, F-147
9	T.H. 201	F-149, F-150
10	696	0-10, 10-20
11	696	20-30, 30-40
12	696	40-50, 50-60, 60-70
12A	696	50-60, 60-70
13	683	0-10, 10-20, 20-30, 30-40
14	683	50-60, 60-70, 70-80, 80-90
14A	683	50-60, 60-70, 70-80
15	685	0-10, 10-20
16	685	20-30, 30-40
17	585	40-50, 50-60, 60-70, 70-80, 90-100
17A	685	40-50, 60-70, 70-80
18	665	0-10, 10-20, 20-30, 30-40
19	665	40-50
20	665	50-60, 60-70
21	665	70-80, 80-90, 100-110, 110-120
21A	665	70-80, 100-110, 110-120
22	665	0-10, 10-20, 20-30, 30-40 40-50, 60-70
22A	678	0-10, 10-20, 20-30, 30-40, 40-50
23	660	0-10, 10-20
24	660	20-30, 30-40, 40-50
24A	660	20-30, 30-40
25	660	50-60, 60-70
26	675	0-10, 10-20, 20-30
27	675	30-40, 40-50, 50-60, 60-70, 70-80
28	675	80-90, 90-100
28A	675	90-100
29	715	0-10, 10-20
30	715	20-30, 30-40, 40-50
31	715	50-60, 60-70, 70-80, 80-90
32	213	0-10, 10-20
33	213	20-30, 30-40, 40-50
34	213	50-60, 60-70, 70-80
34A	213	60-70, 70-80
35	213	80-90, 90-100
36	213	100-110, 110-120, 120-130

TABLE III CHEMICAL ANALYSIS OF LEACHATE SOLUTIONS, SAMPLES 1-36

Sample No.	$\text{HCO}_3^-$ <u>mg/l</u>	$\text{CO}_3^{=}$ <u>mg/l</u>	$\text{SO}_4^{=}$ <u>mg/l</u>	$\text{Cl}^-$ <u>mg/l</u>	$\text{NO}_3^-$ <u>mg/l</u>	$\text{Na}^+$ <u>mg/l</u>	$\text{K}^+$ <u>mg/l</u>	$\text{Ca}^{++}$ <u>mg/l</u>	$\text{Mg}^{++}$ <u>mg/l</u>
1	163.64	---	16.60	6.71	33.39	16.33	2.08	39.96	18.95
2	43.36	---	654.58	3.67	10.15	24.35	8.77	148.27	74.64
3	87.62	---	187.04	6.50	6.21	11.34	7.91	58.29	26.74
4	96.62	---	9.93	1.54	8.84	5.39	5.28	15.19	11.98
5	55.25	---	791.59	8.47	7.01	38.81	9.38	159.08	94.57
6	77.99	---	148.10	3.68	12.07	13.54	3.81	42.56	22.85
7	88.49	---	5.99	2.31	12.25	.96	4.16	26.33	6.88
8	72.14	---	29.45	3.03	3.14	7.40	2.48	16.92	9.72
9	94.64	---	49.92	3.60	5.00	6.72	6.17	26.65	13.06
10	104.22	---	37.44	1.43	2.42	4.99	7.76	27.98	10.40
11	105.62	---	65.73	0.63	1.21	6.54	7.13	35.35	14.44
12	75.53	---	317.35	1.44	1.99	72.44	4.52	43.91	35.73
13	137.89	---	83.20	2.20	4.00	27.84	9.16	30.28	19.54
14	150.77	---	416.00	2.00	1.65	144.31	8.47	48.82	29.89
15	89.71	---	274.56	3.38	3.40	60.47	7.15	37.21	33.59
16	88.74	---	29.95	1.61	2.56	12.82	4.34	17.66	9.55
17	66.09	---	181.71	1.74	1.53	30.59	7.19	44.89	20.66
18	84.58	---	24.96	1.00	1.90	9.19	4.46	16.71	9.11
19	71.51	---	362.75	.46	.60	4.88	7.12	89.68	43.99
20	82.06	---	85.86	.29	3.28	8.40	13.23	26.90	14.80
21*	161.79	---	180.96	3.13	3.27	34.03	17.53	51.55	26.55
22	125.99	---	134.78	3.29	4.09	53.53	16.28	24.13	21.10
23	47.18	---	673.92	5.95	4.61	104.47	17.26	74.95	101.11
24	433.65	---	332.80	6.94	6.18	120.54	20.05	85.92	68.05
25	96.23	---	479.23	1.21	1.34	62.56	16.85	81.29	58.09
26	105.97	---	1431.04	21.28	16.44	233.20	34.16	130.78	178.08
27	154.40	---	126.46	1.85	1.68	31.46	15.74	41.17	21.15
28	82.64	---	252.72	2.13	2.34	67.19	13.33	44.45	29.90
29	111.96	---	15.48	1.05	2.20	7.65	10.48	23.95	10.01
30	172.01	---	126.46	1.87	2.34	31.95	17.12	48.14	21.63
31	162.24	---	227.14	1.46	2.12	75.69	10.95	48.16	24.63
32*	57.58	15.13	42.93	3.01	3.85	41.86	12.47	5.94	4.94
33*	127.21	24.74	119.81	1.17	2.11	121.36	13.95	4.84	3.18
34*	252.69	21.18	74.88	19.23	9.89	178.24	6.61	6.49	1.68
35*	66.59	70.72	25.96	5.59	4.08	109.87	1.79	5.41	1.64
36*	183.77	15.94	164.02	.32	1.87	151.14	1.76	---	2.45

\* - static leached

TABLE IV MEASURED PARAMETERS FOR SOIL LEACHATES, SAMPLES 1-36

<u>Sample No.</u>	<u>gr Soil</u>	<u>Vol liter</u>	<u>pH</u>	<u>S.C. mmhos/cm</u>	<u>Σcation meq/l</u>	<u>Σanion meq/l</u>	<u>TDS mg/l</u>	<u>mg /gr Soil</u>
1	50.0	.0845	8.09	.373	4.32	3.76	297.66	.503
2	50.0	.3885	7.74	1.210	14.80	14.60	967.79	7.520
3	50.0	.1271	7.70	.575	5.81	5.62	391.65	.996
4	50.0	.2776	8.23	.208	2.11	1.97	154.77	.859
5	50.0	.2025	7.87	1.469	17.65	17.75	1164.16	4.716
6	50.0	.2019	8.20	.469	4.69	4.66	324.60	1.311
7	50.0	.2107	8.10	.194	2.03	1.84	147.37	.621
8	50.0	.2071	8.20	.202	2.03	1.93	144.28	.598
9	50.0	.1741	8.20	.298	2.85	2.77	205.76	.716
10	50.0	.2167	8.10	.276	2.67	2.57	196.64	.852
11	50.0	.3578	7.70	.348	3.42	3.14	236.65	1.694
12	46.2	.2515	6.62	.774	8.39	7.92	552.91	3.265
13	50.0	.1304	8.18	.422	4.56	4.12	314.11	.819
14	50.0	.2600	7.10	1.069	11.40	11.22	801.91	4.170
15	50.0	.1792	8.15	.688	7.43	7.48	514.47	1.844
16	50.0	.2015	8.10	.221	2.34	2.16	167.23	.674
17	50.0	.3062	7.22	.528	5.45	4.94	354.40	2.170
18	50.0	.1953	8.15	.202	2.09	1.97	151.91	.593
19	50.0	.3657	7.10	.709	8.49	8.75	580.99	4.249
20	50.0	.2586	7.98	.325	3.26	3.19	234.82	1.214
21*	26.4	.1904	7.57	.597	6.68	6.56	478.81	3.453
22	50.0	.2007	7.99	.551	5.69	5.03	383.19	1.538
23	50.0	.2427	8.03	1.353	17.10	15.10	1029.45	4.997
24	50.0	.0767	8.25	1.254	15.60	14.30	1074.13	1.648
25	50.0	.2078	7.10	1.007	11.90	11.60	796.80	3.312
26	50.0	.0896	8.05	2.479	32.20	32.40	2150.95	3.855
27	50.0	.1763	7.98	.520	5.57	5.24	393.91	1.389
28	50.0	.1574	7.02	.726	7.94	6.72	494.70	1.557
29	50.0	.1387	8.09	.243	2.62	2.22	182.78	.689
30	50.0	.1444	8.11	.558	6.01	5.55	422.02	1.219
31	50.0	.2756	7.32	.729	8.00	7.47	552.39	3.045
32*	50.0	.6157	9.45	.206	2.84	2.49	187.71	2.311
33*	50.0	.3600	9.42	.529	6.14	5.47	418.37	3.012
34*	19.8	.1700	9.08	.746	8.38	7.11	570.89	4.902
35*	50.0	.4262	10.00	.397	4.21	5.23	291.65	2.486
36*	50.0	.3600	9.03	.635	6.82	7.00	521.27	3.753

\* - static leached.

Leaching of samples PR 1 - PR 15 involved mixing of 50.0 gr of air dried soil that had passed through 1 mm sieve with 450 ml of water. The suspension was shaken periodically for 48 hours, centrifuged, and filtered through 0.45  $\mu\text{m}$  filter. The filtrate was split one portion being analyzed for major components, another being preserved for subsequent boron analysis, and the third was analyzed for dissolved organic carbon. Preservation for boron involved acidification of 45 ml of the sample with 1.1 ml of concentrated nitric acid.

The major component analyses and related parameters are listed in Tables V and VI.

Samples PR 4 and PR 9 were leached a total of three times. After the first leach, the soil was dried with a heat lamp and subsequently mixed with 450 ml of water. The suspension was periodically shaken for 48 hours, centrifuged, filtered and analyzed. The procedure was then repeated again. The results are included in Tables V and VI.

#### Trace Component Leaching

The air dry soil samples were leached with diethylene-triamine pentacetic acid to obtain available trace metals. The procedure was similar to that developed by Lindsay and Novell, Agron. Abstr. p. 84, 1969. Ten grams of soil were mixed with 40 ml of DTPA solution (0.01 M  $\text{CaCl}_2$ , 0.1 M triethanol amine, and 0.005 M DTPA), shaken for 2.0 hours and analyzed by atomic absorption. The concentrations of available Cd, Cu, Fe, Mn, Ni, Pb and Zn are summarized in Table VII.

A portion of the obtained extract solution was preserved for mercury analysis by adding sufficient  $\text{HNO}_3$  and  $\text{Cr}_2\text{O}_7^{2-}$  to provide 5% (V/V)  $\text{HNO}_3$  and 0.05%  $\text{Cr}_2\text{O}_7^{2-}$  (C. Feldman, Anal. Chem., 46, 99 (1974)).\*

These samples, along with a portion of the major component leachate, were sent to Dr. Krawczyk, U.S. Environmental Protection Agency, Corvallis, Oregon, for boron and mercury analysis.

#### Leachable Carbon

A portion of the major component leachate solution was analyzed for total dissolved organic carbon using an oceanography total carbon system. The values are presented in Table VIII. The phenolic content was determined using a procedure developed by H. deHaan. (H. deHaan, Proc. Int. Meet. Humic Sub., p. 53-62, 1972, Centre for Agricultural Publishing and Documentation, Wageningen, The Netherlands, 1975). The procedure involves mixing 2.0 ml of water sample with 7 ml of 1%  $\text{Na}_2\text{CO}_3$  in 0.1 N  $\text{NaOH}$ , and 0.5 ml of Folin and Ciocalteau's phenol reagent-2 normal. The colour is developed for 30 minutes at 21°C and monitored at 720 nm. The working range extends from 1-100 mg/l

\* For dilution correction multiply by 1.21.

TABLE V CHEMICAL ANALYSIS OF LEACHATE SOLUTIONS, PR 1 - PR 15

Values are mg/g Soil

Sample No.	<u>HCO<sub>3</sub><sup>-</sup></u>	<u>CO<sub>3</sub><sup>=</sup></u>	<u>SO<sub>4</sub><sup>=</sup></u>	<u>Cl<sup>-</sup></u>	<u>NO<sub>3</sub><sup>-</sup></u>	<u>Na<sup>+</sup></u>	<u>K<sup>+</sup></u>	<u>Ca<sup>++</sup></u>	<u>Mg<sup>++</sup></u>
PR 1	.479	--	.067	.018	.179	.021	.053	.133	.061
PR 2	.250	--	1.380	.023	.069	.077	.018	.259	.213
PR 3	.220	--	.118	.017	.044	.025	.028	.057	.020
PR 4	.267	--	4.178	.036	.162	.168	.034	1.180	.445
PR 4(2)	.243	--	.521	.004	.006	.021	.009	.154	.062
PR 4(3)	--	--	--	--	--	--	--	--	--
PR 5	.335	--	.027	.013	.004	.061	.032	.064	.011
PR 6	.382	.021	.017	.006	.010	.022	.015	.060	.065
PR 7	.222	.002	1.302	.009	.006	.093	.017	.273	.194
PR 8	.161	--	5.812	.030	.025	.335	.018	1.319	.715
PR 9	.226	--	4.268	.031	.033	.189	.021	.955	.542
PR 9(2)	.249	.026	.318	.003	--	.024	.006	.088	.067
PR 9(3)	--	--	--	--	--	--	--	--	--
PR10	.234	--	1.784	.003	.009	.070	.016	.383	.240
PR11	.387	.003	.005	.007	.019	.008	.016	.105	.026
PR12	.258	.008	.083	.003	--	.024	.005	.054	.044
PR13	.313	--	1.078	.021	.006	.118	.027	.240	.129
PR14	.165	--	1.797	.014	.022	.127	.014	.523	.239
PR15	.185	--	2.695	.036	.025	.189	.025	.609	.391

TABLE VI MEASURED PARAMETERS FOR SOIL, PR 1 - PR 15

50 gr Soil + 450 ml H<sub>2</sub>O

Sample No.	pH	S.C. mmhos/cm	<u>Σcation meq/l</u>	<u>Σanion meq/l</u>	<u>mg/gr Soil</u>
PR 1	7.87	.162	1.54	1.41	1.01
PR 2	8.02	.413	3.81	3.85	2.29
PR 3	7.60	.074	0.69	0.81	0.53
PR 4	7.87	1.026	11.50	10.60	6.47
PR 4(2)	8.30	.168	1.59	1.67	1.03
PR 4(3)	--	.006	--	--	--
PR 5	8.00	.092	0.85	0.72	0.55
PR 6	8.80	.102	1.07	0.85	0.60
PR 7	8.39	.388	3.79	3.47	2.12
PR 8	7.88	1.281	15.50	13.90	8.42
PR 9	8.10	.983	11.20	10.40	6.27
PR 9(2)	9.01	.126	1.22	1.30	0.78
PR 9(3)	--	.006	--	--	--
PR10	8.15	.466	4.69	4.58	2.74
PR11	8.38	.086	0.91	0.79	0.58
PR12	8.54	.079	0.83	0.70	0.48
PR13	7.99	.326	3.16	3.14	1.93
PR14	8.12	.564	5.74	4.54	2.90
PR15	8.19	.730	7.94	6.74	4.16

TABLE VII TRACE METALS EXTRACTED FROM POPLAR RIVER SOIL  
AND FLY ASH, VALUES in  $\mu\text{g/g}$  OF AIR DRY SOIL

Sample No.	Cd	Cu	Fe	Mn	Ni	Pb	Zn	B
29	0.116	2.84	32.60	9.48	1.10	2.24	1.30	0.29
30	0.116	5.21	54.40	30.60	1.10	3.58	2.36	0.09
31	0.124	7.72	155.60	98.80	2.01	6.18	22.12	13.67
32	<.080	2.56	58.82	10.28	<.40	2.40	1.57	0.32
33	0.116	5.76	107.52	27.73	1.39	2.84	8.14	0.78
34A	0.144	1.53	30.34	1.65	3.95	5.58	11.69	2.43
35	<.080	0.33	27.65	1.14	<.40	3.28	4.13	0.60
36	0.120	7.68	65.66	10.14	2.17	9.47	12.31	5.21
PR1	<.080	1.56	16.96	7.00	1.20	<1.00	6.04	
PR2	<.080	1.92	6.60	3.84	2.00	<1.00	0.44	0.82
PR3	<.080	1.52	9.68	7.96	1.60	<1.00	2.52	0.32
PR4	<.080	2.68	16.04	4.88	0.40	1.04	0.88	0.70
PR5	<.080	1.04	3.96	20.36	2.00	<1.00	0.32	0.65
PR6	<.080	3.68	4.28	2.03	<.40	<1.00	1.04	0.40
PR7	0.088	1.64	5.04	1.12	2.00	<1.00	0.20	0.87
PR8	0.080	1.56	7.04	0.84	2.00	<1.00	0.52	1.37
PR9	<.080	1.36	4.56	0.96	<.40	<1.00	0.12	0.93
PR10	<.080	2.00	3.32	1.16	2.40	<1.00	0.24	0.64
PR11	<.080	1.28	0.84	1.60	<.40	<1.00	0.16	0.23
PR12	<.080	1.28	4.88	2.12	<.40	<1.00	0.28	0.38
PR13	0.100	1.56	13.68	5.20	<.40	<1.00	0.64	0.69
PR14	<.080	1.16	1.92	1.08	<.40	<1.00	0.36	0.54
PR15	<.080	1.08	3.68	2.08	<.40	<1.00	0.44	0.84

TABLE VIII LEACHABLE CARBON

<u>Sample No.</u>	<u>DOC mg/L</u>	<u>Abs</u>	<u><math>10^3</math> Abs/DOC</u>	<u>mgC/gr Soil</u>
1	21.1	.019	0.90	0.211
2	8.2	.003	0.37	0.082
3	9.2	.011	1.20	0.092
4	18.0	.018	1.00	0.180
5	8.5	.006	0.76	0.085
6	7.5	.009	1.20	0.075
7	19.9	.020	1.01	0.199
8	5.6	.010	1.79	0.056
9	4.4	.014	3.21	0.044
10	5.4	.014	2.59	0.054
11	6.1	.026	4.26	0.061
12A	27.3	.065	2.38	0.273
13	5.7	.015	2.61	0.057
14A	21.4	.053	2.48	0.214
15	6.5	.010	1.54	0.065
16	5.8	.011	1.90	0.058
17A	27.0	.037	1.35	0.270
18	14.4	.017	1.22	0.144
19	17.9	.029	1.65	0.179
20	30.9	.011	0.34	0.309
21A	25.4	.039	1.54	0.254
22A	14.4	.018	1.22	0.144
23	14.5	--	--	0.145
24A	11.7	.012	1.03	0.117
25	13.7	.046	3.36	0.137
26	7.1	.009	1.28	0.071
27	5.9	.017	2.86	0.059
28A	13.1	.036	2.71	0.131
29	6.9	.012	1.73	0.069
30	8.5	.027	3.17	0.085
31	19.9	.040	2.01	0.199
32	6.6	.004	0.61	0.066
33	10.4	.008	0.82	0.104
34A	13.6	.027	2.02	0.136
35	6.2	.026	4.19	0.062
36	19.8	.029	1.49	0.198

tyrosine using a 1 cm cell. An absorbance of 0.15 is equivalent to 10 mg/l of tyrosine. The values are listed in Table VIII.

### Quality Control

Table X lists the results of standard additions to the leachate solutions. No comparable data is available for the trace metal determination. However, the sensitivity and reproducibility has been studied by Lindsay and our values should be comparable. The total organic carbon analysis has a standard deviation of 10.5 mg/l.

### Conclusions

A large range in amount of readily leachable salts is observed for the soils. For comparison purposes the mg/g values have been converted to Kg/m<sup>3</sup> using an assumed bulk density of 1.5 g/cc. The values are listed in Table X.

There are several soil samples that contain large amounts of readily leachable salts. These are 2, 5, 23, 34, PR 4, PR 8, and PR 9. The available sulfate is higher in samples 2, 5, 23, PR 4, PR 8, and PR 9.

The successive leaching of samples PR 4 and PR 9 indicate that more than 85% of the leachable salts are removed with the first leach and the second takes out a majority of what is left.

The amount of organic carbon that is leached from these soils is low and comparable to that observed in subsurface soils collected from the Fort Union region. Using the guideline that an absorbance of 0.15 is equivalent to 10 mg/l tyrosine, these samples could contain up to 3 mg/l phenol as tyrosine. The sensitivity for the phenol measurement is approximately 0.5 mg/l tyrosine.

The State of Montana has identified levels of DTPA extractable trace metals that may indicate potential trace metal toxicity:

- Cd: Values in the 0.1 to 1.0 µg/g may cause problems.
- Cu: The normal values range from 0.5 - 40 µg/g. Values about 40 may cause problems.
- Fe: Soils containing 1.5 - 2% total iron and DTPA values in excess of 200 µg/g are often observed. No hazard would be expected at this level.
- Mn: Normal values of extractable Mn range from 0.5 - 40 µg/g.
- Ni: Levels of DTPA extractable above 1 µg/g may have toxic effects on plants.

The results of trace metal analysis of water leached from the mine spoil and reservoir soil samples is provided in Table XI.

TABLE IX STANDARD ADDITION STUDIES

<u>Element</u>	<u>Water Sample mg/L</u>	1st Addition			2nd Addition		
		<u>Amt. Added</u>	<u>Found</u>	<u>%</u>	<u>Amt. Added</u>	<u>Found</u>	<u>%</u>
Ca <sup>2+</sup>	67.70	68.60	136.00	-0.2	46.20	112.60	-1.1
Mg <sup>2+</sup>	35.40	46.35	84.00	+2.8	82.80	127.30	+7.7
Na <sup>+</sup>	4.70	11.90	15.70	-5.4	22.00	26.60	-0.4
K <sup>+</sup>	4.54	44.70	49.70	+0.9	21.00	25.60	+0.2
SO <sub>4</sub> <sup>2-</sup>	300.00	164.00	468.00	+0.9	111.00	414.00	+0.7
NO <sub>3</sub> <sup>-</sup>	4.10	28.80	32.50	-1.2	60.00	66.50	+3.7
CL <sup>-</sup>	5.40	40.50	44.30	-3.5	19.00	22.10	-9.4
HCO <sub>3</sub> <sup>-</sup>	35.80	31.70	73.50	+8.9	58.40	98.40	+4.5

TABLE X KILOGRAMS OF READILY SOLUBLE SALTS PER CUBIC METER

<u>Sample No.</u>	<u>kg/m<sup>3</sup></u>	<u>Sample No.</u>	<u>kg/m<sup>3</sup></u>	<u>Sample No.</u>	<u>kg/m<sup>3</sup></u>
1	0.75	21	5.18	PR 4(2)	1.55
2	11.28	22	2.31	PR 4(3)	--
3	1.50	23	7.50	PR 5	0.83
4	1.29	24	2.47	PR 6	0.90
5	7.08	25	4.97	PR 7	3.18
6	1.96	26	5.78	PR 8	12.63
7	0.93	27	2.08	PR 9	9.41
8	0.90	28	2.34	PR 9(2)	1.17
9	1.07	29	1.03	PR 9(3)	--
10	1.28	30	1.83	PR10	4.11
11	2.54	31	4.57	PR11	0.87
12	4.90	32	3.47	PR12	0.72
13	1.23	33	4.52	PR13	2.90
14	6.26	34	7.35	PR14	4.35
15	2.77	35	3.73	PR15	6.24
16	1.01	36	5.63		
17	3.26	PR 1	1.52		
18	0.89	PR 2	3.44		
19	6.37	PR 3	0.80		
20	1.82	PR 4	9.71		

TABLE XI TRACE METALS WATER LEACHED FROM SOIL AND FLY ASH SAMPLES  
Values are  $\mu\text{g/g}$

Sample No.	Cd	Cr	Cu	Fe	Mn	Ni	Pb	Zn
1	--	--	0.14	2.07	0.06	--	--	0.12
2	0.06	0.08	0.18	0.54	0.07	0.37	--	1.31
3	0.05	--	0.14	0.38	0.13	0.09	0.09	0.58
4	--	--	0.14	0.62	0.03	--	--	0.15
5	0.06	0.07	0.13	0.62	0.06	0.09	--	0.25
6	0.05	0.01	0.14	0.41	0.11	--	--	0.12
7	0.01	--	0.06	2142	0.10	--	--	0.13
8	0.01	--	0.06	0.44	0.02	--	--	0.09
9	0.01	--	0.03	0.77	0.21	--	--	0.06
10	0.01	--	0.14	0.53	0.11	--	--	0.06
11	0.02	0.06	0.05	0.33	0.47	--	--	0.08
12A	--	0.06	0.05	0.50	1.19	--	--	0.22
13	--	0.06	0.11	0.99	0.13	--	--	0.10
14A	0.02	0.06	0.05	0.33	0.47	--	--	0.08
15	--	--	0.02	0.28	0.01	--	--	0.12
16	0.02	--	0.07	1.18	0.02	--	--	0.10
17	0.02	0.04	0.07	0.42	0.52	0.28	0.18	0.16
18	0.01	0.01	0.06	1.11	0.05	0.18	--	0.15
19	0.01	0.04	0.01	0.53	1.80	0.09	--	0.22
20	0.03	0.06	0.06	0.49	0.35	0.09	0.18	0.05
21A	0.03	0.08	0.14	1.62	0.81	0.18	0.37	0.12
22A	0.02	0.05	0.10	0.52	0.07	0.09	0.37	0.06
23	0.05	0.08	0.10	0.60	0.05	0.18	0.37	0.17
24A	0.03	0.07	0.06	1.01	0.07	0.09	--	0.18
25	0.03	0.06	0.09	0.33	1.04	0.09	--	0.18
26	0.02	0.06	0.05	0.43	0.10	0.09	--	0.12
27	0.01	0.04	0.03	0.45	0.31	--	--	0.06
28A	0.03	0.06	0.09	0.72	0.18	0.09	0.46	0.12
29	0.01	0.01	0.06	1.28	0.05	--	--	0.05
30	0.03	0.03	0.07	1.75	0.30	0.09	--	0.05
31	0.01	0.03	0.09	1.63	1.37	0.09	0.46	0.19
32	0.02	0.04	0.13	5.24	0.10	0.09	0.46	0.16
33	0.03	0.10	0.13	2.52	0.06	0.18	0.55	0.11
34A	0.08	0.15	0.21	1.14	0.06	0.28	0.74	0.05
35	0.01	--	0.01	0.33	0.02	--	--	0.08
36	0.01	0.01	0.05	0.51	0.21	--	--	0.51
PR- 1								
PR- 2	--	0.06	0.03	0.77	0.02	0.09	--	0.09
PR- 3	0.01	0.03	0.08	1.63	0.03	--	0.18	0.21
PR- 4	0.02	0.06	0.01	0.36	0.04	0.09	--	0.08
PR- 4(2)	--	0.67	--	1.05	--	--	--	--
PR- 5	--	0.01	0.06	4.43	0.07	--	--	0.39
PR- 6	0.02	2.20	0.37	6.50	0.05	0.09	10.23	5.33
PR- 7	--	0.05	0.03	0.28	0.01	--	--	0.11
PR- 8	0.02	0.09	0.08	0.37	0.02	0.09	--	0.18
PR- 9	0.03	0.32	0.08	0.46	0.01	0.09	1.57	0.55
PR- 9(2)	--	--	--	0.38	--	--	--	0.09
PR-10	0.02	0.03	0.01	0.77	0.02	--	--	0.06
PR-11	--	--	0.03	0.73	0.01	--	0.18	0.04
PR-12	--	--	0.01	0.36	0.01	--	0.18	0.04
PR-13	0.02	0.19	0.11	0.34	0.02	0.09	--	0.22
PR-14	--	--	--	0.19	--	--	--	0.05
PR-15	--	.75	--	1.03	0.06	--	--	0.16

TABLE XI continued. Values are  $\mu\text{g/g}$ 

Sample No.	As	Mo	Sr	Se	Zr	Sb	Co	Al	V
1	--	--	0.30	--	0.03	--	--	1.84	0.02
2	0.09	--	4.87	--	0.06	0.18	--	2.95	--
3	0.09	--	1.10	--	0.06	--	--	0.83	0.02
4	0.09	--	0.30	--	0.04	0.18	--	0.55	0.02
5	0.09	--	3.04	--	0.04	0.09	--	1.94	--
6	--	--	0.80	--	0.06	0.18	--	0.74	--
7	0.09	--	0.19	--	--	--	--	2.86	--
8	0.09	--	0.30	0.09	--	0.09	0.28	0.83	--
9	0.09	--	0.50	--	--	--	--	0.55	--
10	--	--	0.39	0.09	0.01	0.37	--	0.65	0.01
11	--	--	1.35	--	0.01	0.09	--	0.83	0.01
12A	--	--	2.68	0.09	--	--	--	0.74	--
13	--	--	0.59	0.09	--	0.09	--	0.83	--
14A	--	--	1.88	--	--	--	--	0.65	--
15	--	--	0.85	--	--	--	--	0.18	--
16	0.18	--	0.39	--	--	--	--	0.83	--
17	--	--	4.66	--	--	--	--	0.92	0.02
18	--	--	0.45	0.18	--	--	--	0.83	--
19	--	--	3.30	--	--	--	--	1.48	--
20	0.37	--	1.10	0.18	0.06	0.18	--	0.92	0.06
21A	0.37	1.48	2.89	0.18	0.06	0.18	0.55	1.66	0.10
22A	0.18	0.92	0.64	0.18	0.04	0.18	0.28	0.74	0.04
23	0.37	--	2.83	0.18	0.06	0.18	--	1.75	0.06
24A	0.18	--	1.01	0.09	0.04	0.18	--	1.38	0.06
25	0.18	--	2.91	0.18	0.04	0.09	--	1.11	0.04
26	--	--	2.78	--	0.02	0.09	0.09	1.20	0.02
27	0.18	--	0.86	--	0.02	--	0.28	0.83	0.03
28A	0.18	0.46	0.74	0.18	0.06	0.18	0.37	1.66	0.10
29	--	--	0.32	--	--	0.09	--	1.11	--
30	--	--	0.80	0.18	0.01	0.18	0.09	1.20	0.04
31	--	--	1.95	0.28	0.01	0.18	0.28	0.92	0.04
32	--	2.21	0.30	0.28	0.06	0.74	0.83	6.55	0.06
33	0.18	--	0.18	0.37	0.07	0.46	--	4.89	0.10
34A	0.92	--	9.12	0.83	0.11	0.83	--	2.95	0.27
35	--	--	4.48	--	--	--	--	0.74	--
36	--	0.18	7.46	0.18	--	--	--	1.48	--
PR- 1									
PR- 2	--	--	1.53	0.18	--	--	0.18	0.83	0.02
PR- 3	--	--	0.16	0.18	--	--	0.18	1.66	0.02
PR- 4	--	0.46	4.46	0.18	--	0.09	0.18	2.21	--
PR- 4A	--	--	0.65	--	--	--	--	0.28	--
PR- 5	--	--	0.18	0.18	--	--	--	5.35	0.02
PR- 6	--	--	0.31	--	--	--	0.18	1.48	0.02
PR- 7	--	--	1.36	--	--	--	--	0.92	--
PR- 8	--	1.11	5.83	0.18	--	0.09	0.18	2.67	--
PR- 9	0.90	1.38	3.51	0.18	0.02	0.09	0.37	2.12	0.02
PR- 9A	--	--	0.50	--	--	--	--	--	--
PR-10	--	0.46	1.47	--	--	--	--	0.92	0.07
PR-11	--	--	0.21	--	--	--	--	0.92	--
PR-12	--	--	0.26	--	--	0.09	0.18	0.18	--
PR-13	--	1.38	1.17	--	--	0.09	0.18	0.83	0.01
PR-14	--	--	1.83	--	--	--	--	0.65	--
PR-15	--	--	0.66	--	--	--	--	2.58	--

TABLE XI continued. Values are  $\mu\text{g/g}$ 

Sample No.	Cd	Cr	Cu	Fe	Mn	Ni	Pb	Zn
1	--	--	0.14	2.07	0.06	--	--	0.12
2	0.06	0.08	0.18	0.54	0.07	0.37	--	1.31
3	0.05	--	0.14	0.38	0.13	0.09	0.09	0.58
4	--	--	0.14	0.62	0.03	--	--	0.15
5	0.06	0.07	0.13	0.62	0.06	0.09	--	0.25
6	0.05	0.01	0.14	0.41	0.11	--	--	0.12
7	0.01	--	0.06	2.42	0.10	--	--	0.13
8	0.01	--	0.06	0.44	0.02	--	--	0.09
9	0.01	--	0.03	0.77	0.21	--	--	0.06
10	0.01	--	0.14	0.53	0.11	--	--	0.06
11	0.02	0.06	0.05	0.33	0.47	--	--	0.08
12A	--	0.06	0.05	0.50	1.19	--	--	0.22
13	--	0.06	0.11	0.99	0.13	--	--	0.10
14A	--	--	--	0.41	1.16	--	--	0.08
15	--	--	0.02	0.28	0.01	--	--	0.12
16	0.02	--	0.07	1.18	0.02	--	--	0.10
17	0.02	0.04	0.07	0.42	0.52	0.28	0.18	0.16
18	0.01	0.01	0.06	1.11	0.05	0.18	--	0.15
19	0.01	0.04	0.01	0.53	1.80	0.09	--	0.22
20	0.03	0.06	0.06	0.49	0.35	0.09	0.18	0.05
21A	0.03	0.08	0.14	1.62	0.81	0.18	0.27	0.12
22A	0.02	0.05	0.10	0.52	0.07	0.09	0.37	0.06
23	0.05	0.08	0.10	0.60	0.05	0.18	0.37	0.17
24A	0.03	0.07	0.06	1.01	0.07	0.09	--	0.18
25	0.03	0.06	0.09	0.33	1.04	0.09	--	0.18
26	0.02	0.06	0.05	0.43	0.10	0.09	--	0.12
27	0.01	0.04	0.03	0.45	0.31	--	--	0.06
28A	0.03	0.06	0.09	0.72	0.18	0.09	0.46	0.12
29	0.01	0.01	0.06	1.28	0.05	--	--	0.05
30	0.03	0.03	0.07	1.75	0.30	0.09	--	0.05
31	0.01	0.03	0.09	1.63	1.37	0.09	0.46	0.19
32	0.02	0.04	0.13	5.24	0.10	0.09	0.46	0.16
33	0.03	0.10	0.13	2.52	0.06	0.18	0.55	0.11
34A	0.08	0.15	0.21	1.14	0.06	0.28	0.74	0.05
35	0.01	--	0.01	0.33	0.02	--	--	0.08
36	0.01	0.01	0.05	0.51	0.21	--	--	0.51
PR-1								
PR-2	--	0.06	0.03	0.77	0.02	0.09	--	0.09
PR-3	0.01	0.03	0.08	1.63	0.03	--	0.18	0.21
PR-4	0.02	0.06	0.01	0.36	0.04	0.09	--	0.08
PR-4(2)	--	0.67	--	1.05	--	--	--	--
PR-5	--	0.01	0.06	4.43	0.07	--	--	0.39
PR-6	0.02	2.20	0.37	6.50	0.05	0.09	10.23	5.33
PR-7	--	0.05	0.03	0.28	0.01	--	--	0.11
PR-8	0.02	0.09	0.08	0.37	0.02	0.09	--	0.18
PR-9	0.03	0.32	0.08	0.46	0.01	0.09	1.57	0.55
PR-9(2)	--	--	--	0.38	--	--	--	0.09
PR-10	0.02	0.03	0.01	0.77	0.02	--	--	0.09
PR-11	--	--	0.03	0.73	0.01	--	--	0.06
PR-12	--	--	0.01	0.36	0.01	--	0.18	0.04
PR-13	0.02	0.19	0.11	0.34	0.02	0.09	--	0.22
PR-14	--	--	--	0.19	--	--	--	0.05
PR-15	--	.75	--	1.03	0.06	--	--	0.16

## COOKSON RESERVOIR LEACHING STUDY

As the level of water in the reservoir rises, the soluble constituents of the flooded surface will be taken into solution. The degree to which this occurs will depend on the quantity of water accumulated during successive spring floods, the depth and wind induced wave action, anaerobic conditions developed under ice cover, and subsequent overturn in the spring melt.

### Soil Sampling and Site Conditions

To assess this leaching situation, soil samples were collected by Sheppard T. Powell Consultants Limited during October and November 1974, from 18 acres in the two valleys which will be flooded. The locations of these sample collection points are shown in Figure 1. A description of the area surrounding each collection point is given below.

All soil samples were an average of the top 6 to 8 inches of surface soil except for sample No. 11. All vegetation was scraped from the surface before the samples were taken.

Sample No. 1 was taken in the centre of the East Poplar River Valley about 200 ft. east of the level crossing on the river. This is on the south boundary of the S.E. 1/4 of section 6, Tp. 2, Rge. 26. Location marked on site by the fence corner.

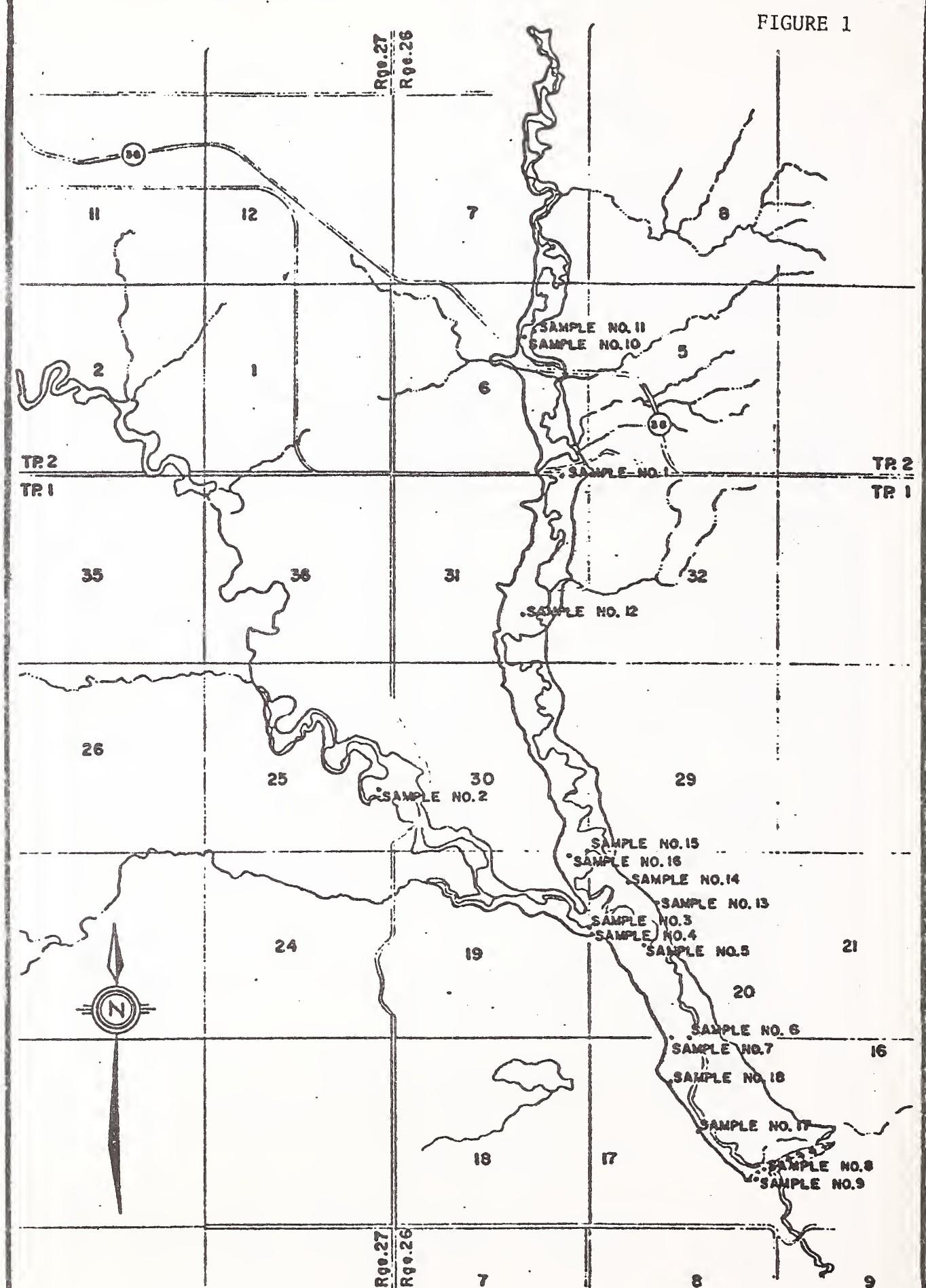
Sample No. 2 was taken in the centre of the Creek Valley feeding into the Poplar River. The area is inside a U-bend in the creek and the closest direction to the creek was about 200 ft. south. This is near the east boundary of the S.E. 1/4 of Sec. 25, Tp. 1, Rge. 27. Marked location on site with a 3 ft. fence picket.

Sample No. 3 was taken from the bottom of the creek feeding the Poplar River. This point was on the east boundary of the N.E. 1/4 of Sec. 19, Tp. 1, Rge. 26.

Sample No. 4 was taken near the centre of the Creek Valley feeding the Poplar River at the junction of the creek and river. The sample was taken near a post that was wrapped with wire planted in the valley floor. This point was on the east boundary of the N.E. 1/4 Sec. 19, Tp. 1, Rge. 26.

Sample No. 5 was taken from the top of the west bank of the Valley floor. This point was about 5'-0" above the water surface alongside the fence. This point is located on the north boundary of the south west 1/4 of Sec. 20, Tp. 1, Rge. 26.

FIGURE 1



# SASKMONT ENGINEERING COMPANY LIMITED

SASKATCHEWAN POWER CORPORATION			
DRN.	M.A.H.	CRD. L.M.Q.	FILE 0326 DATE 22-11-74
POPLAR RIVER POWER PROJECT			SHEET NO. 1
<p style="text-align: center;"><b>SOIL SAMPLE LOCATIONS</b></p>			

Sample No. 6 was taken in the centre of the Valley floor, beside a fence. The point was marked by an old fence post. This point is on the south boundary of the S.E. 1/4 Sec. 20, Tp. 1, Rge. 26.

Sample No. 7 was taken from a boggy area in the floor of the Valley. This area was located at the toe slope of the west side of the Valley. This point is just west of 1/4 line on the south boundary of the S.W. 1/4 Sec. 20, Tp. 1, Rge. 26.

Sample No. 8 was taken in the centre of the Valley directly between two hills and about 50 ft. south of the river. Planted an old post in exact location. This point is located near the centre of the east half of the S.E. 1/4 of Sec. 17, Tp. 1, Rge. 26.

Sample No. 9 was taken from the top of a small hill near a rock pile. This hill is on the west side of the Valley about 25 ft. above the Valley floor. This point is near the centre of the S.E. 1/4 of Sec. 17, Tp. 1, Rge. 26.

Sample No. 10 was taken in the centre of the East Poplar River Valley. The area was rough and the point was about 200 ft. from the river. The exact point was near the centre of the west half of the N.E. 1/4 of Sec. 6, Tp. 2, Rge. 26.

Sample No. 11 was taken from a 5 ft. cut bank on the east side of the Valley. The sample was an average from 4 ft. height to the toe-slope of this bank. The exact point is near the centre of the N.E. 1/4 of Sec. 6, Tp. 2, Rge. 26.

Sample No. 12 was taken from the floor of the Valley near the centre of the Valley. Around the point were several large pot holes between 6 to 8 ft. deep with water in them. The surface level of the water was about 4 ft. below the Valley floor. The exact location was near the centre of the S.E. 1/4 of Sec. 31, Tp. 1, Rge. 26.

Sample No. 13 was taken from the east side slope of the Valley about 30 ft. above the Valley floor. The slope was very gentle and the exact location was marked by a small rock pile. This area has been cultivated and seeded back to grass a few years ago. The point is near the centre of the east half of the N.W. 1/4 of Sec. 20, Tp. 1, Rge. 26.

Sample No. 14 was taken near a fence on the edge of the cultivated area described in sample No. 13. This point is about 10 ft. above the Valley floor. The exact location is near the centre of the north half of the N.W. 1/4 of Sec. 20, Tp. 1, Rge. 26.

Sample No. 15 was taken in the centre of the East Poplar Valley from the top of the west bank of the River. The exact location is the N.E. corner of Sec. 19, Tp. 1, Rge. 26.

Sample No. 16 was taken at the toe-slope of the west side of the Valley. This sample was taken on the edge of a cultivated field which had been cropped this year. The exact location was on the north boundary, 1/8 of a mile west of the N.E. corner of Sec. 19, Tp. 1, Rge. 26.

Sample No. 17 was taken from the west side-slope of the East Poplar River Valley about 20 ft. above the Valley floor. The sample was collected on the east west 1/4 line alongside a fence 3/8 of a mile west of the east section line in virgin prairie. The exact location is on the south boundary 1/8 of a mile east of the south west corner of the N.E. 1/4 of Sec. 17, Tp. 1, Rge. 26.

Sample No. 18 was taken from the west side-slope of the East Poplar River Valley about 15 ft. above the toe-slope of the Valley floor. This was just north across a ravine from an old foundation. The point was on the edge of a cultivated field that had been summer-fallowed in the past year. The exact location was near the centre of the east half of the N.E. 1/4 of Sec. 17, Tp. 1, Rge. 26.

#### Results of Leaching Analysis

Each sample was mixed and reduced to a representative 50 grams, which was suspended in 2 litres of demineralized water and stirred for 30 minutes at ambient temperature. The suspension was allowed to settle overnight, and the supernatant liquor decanted, centrifuged and filtered prior to being subjected to standard water analysis.

The data for each point were weighted in terms of the area represented (where two sample points were close together, they were averaged). The weightings are given below.

<u>Sample #</u>	<u>Weighting</u>
1	2
2	2
3,4	2
5	1
6,7	1
8,9	1
10,11	2
12	3
13,14	1
15,16	1
17	1
18	1

The analytical data for the leaching experiments is given in Table 1.

At the 751.3 m datum level, the total volume of the reservoir is  $30.2 \times 10^6 \text{m}^3$  with an area of  $5.75 \times 10^6 \text{m}^2$ . It has been assumed that

only the upper 15 cm layer of soil will, on average, be seriously disturbed during the filling period. This is equivalent to a 1:40 suspension of the soil in water, the concentration at which the laboratory tests were done. The results are given in column 3 of Tables 2 and 3 for the general chemical parameters, and for nutrients and metals respectively.

#### References

Saskmont Engineering Company Limited, Plant Concept, Poplar River Power Station, prepared for Saskatchewan Power Corporation, Volume 3, pp. 1-20 - 1-23 (by Sheppard T. Powell Consultants Limited), 1975.

TABLE 1

Composition of 40:1 Leachate from Poplar Reservoir Soil Samples  
 Saskmont Engineering Co. Ltd. / Sheppard T. Powell Consultants Limited  
 Samples Collected October 22nd - November 5th, 1974

	SAMPLE COLLECTION POINT																		
	PPM																		
pH	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	
Alkalinity-P	8.4	7.3	7.4	7.6	9.0	8.9	8.7	8.7	8.35	8.80	8.80	8.80	8.80	8.80	8.80	8.80	8.90	6.95	
Alkalinity-T	Caco <sub>3</sub>	4	-	-	-	20	6	6	-	8	16	14	-	-	-	-	-	-	
Conductivity	Caco <sub>3</sub>	74	18	20	22	24	70	200	60	30	60	70	118	6	42	42	8	11	
Total Dissolved Solids	umhos	11.2	34	365	53	65	385	142	131	61	152	225	180	14	72	238	85	20	
Sulphate	SO <sub>4</sub>	29	30	120	25	9	85	34	23	14	44	75	56	10	13	103	20	28	
Chloride	Cl	0.2	L0.1	0.4	0.2	1.6	0.8	L0.1	L0.1	0.9	2.5	1.1	1.1	0.6	1.0	1.1	0.8	1.1	
Silica	SiO <sub>2</sub>	3.7	7.1	2.8	6.8	6.9	10.0	7.6	6.8	7.9	11.7	5.7	20.3	5.2	8.1	11.0	9.0	11.0	
Calcium	Ca	7	3	47	3	5	7	7	10	4	8.0	2.4	7.2	2.4	9.6	8.0	13.6	5.6	
Magnesium	Mg	5	2	11	2	3	5	10	4	2	7.8	2.9	20.4	1.5	3.9	4.4	4.9	2.4	
Sodium	Na	15	1	3	4	4	57	19	15	6	17	54	38	0.5	0.6	54	1.8	0.5	
Potassium	K	6	3	1	4	5	7	8	3	3	5.0	1.4	12.1	1.7	3.5	5.9	4.6	1.7	
Total Hardness EDTA	Caco <sub>3</sub>	37	16	163	16	24	35	61	40	18	52	18	102	12	40	38	54	24	
<u>NUTRIENTS</u>																			
Phosphate-Ortho	P	0.04	0.04	0.01	0.03	0.02	0.08	0.09	0.03	0.06	0.05	0.04	0.11	0.05	0.02	0.03	0.02	0.08	
Phosphate-Total	P	0.22	0.25	0.11	0.12	0.07	7.10	0.03	0.15	0.26	0.24	0.11	0.17	0.05	0.12	0.04	0.02	0.09	
Nitrate-Total	N	0.12	0.15	0.03	0.11	0.08	0.11	0.25	0.12	0.17	0.17	0.31	0.14	0.10	0.08	0.25	0.32	0.12	
Ammonia	N	1.4	1.3	0.9	2.0	1.7	2.1	2.3	1.9	1.8	1.6	1.7	2.6	1.0	1.1	0.6	1.0	1.2	
C.O.D.	O <sub>2</sub>	28	2	35	21	14	48	37	20	29	44	36	63	1.0	6.8	12	3.3	1.0	
<u>HEAVY METALS (EXTRACTABLE)</u>																			
Iron	Fe	4.40	1.22	0.32	1.40	0.70	2.50	9.30	3.60	0.40	4.8	2.8	6.7	0.5	0.7	4.1	0.6	0.4	
Copper	Cu	0.07	0.07	0.07	0.05	0.05	0.07	1.0.01	0.02	1.0.01	1.0.01	0.01	1.0.01	1.0.01	1.0.01	1.0.01	1.0.01	1.0.01	
Zinc	Zn	0.11	0.05	0.01	0.01	0.04	0.05	0.14	0.23	0.20	0.30	0.01	0.04	0.01	0.10	0.07	0.16	0.01	
Manganese	Mn	0.37	0.18	0.18	0.07	0.07	1.12	1.85	0.34	0.13	0.25	1.0.05	0.20	0.05	0.10	1.0.05	0.05	0.05	
Lead	Pb	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Mercury	ug/l Hg	-	-	-	-	-	-	-	-	-	1.28	0.92	1.03	0.23	0.09	0.07	1.0.05	0.06	0.11

TABLE 2

Concentration of Inorganic Constituents on  
Completion of Reservoir Fill

		Weighted Average 1973	Weighted Average 1974	Increment From Soil Leaching	Range On Completion of Reservoir Fill
pH		8.25	7.8	8.0	7.9 - 8.1
P. Alkalinity	ppm CaCO <sub>3</sub>	4	2	5	7 - 9
T. Alkalinity	ppm CaCO <sub>3</sub>	452	113	57	170 - 509
Conductivity	umhos	1,118	275	128	403 - 1,246
Turbidity	FTU	5	77	-	5 - 77
Colour	APHA	8	50	-	8 - 50
Total Dissolved Solids ppm		748	192	136	328 - 884
Sulphate	ppm SO <sub>4</sub>	176	38	46	84 - 222
Chloride	ppm Cl	1.6	0.8	0.8	1.6 - 2.4
Silica	ppm SiO <sub>2</sub>	4.4	5.4	9.3	13.7 - 14.7
Calcium	ppm Ca	57	18	8	26 - 65
		(143)	(46)	(21)	(67) - (164)
Magnesium	ppm Mg	49	11	7	18 - 56
		(203)	(45)	(27)	(72) - (230)
Sodium	ppm Na	126	24	17	41 - 143
		(274)	(53)	(37)	(90) - (311)
Potassium	ppm K	12	9	4	13 - 16
		(15)	(11)	(5)	(16) - (20)
Total Hardness	ppm CaCO <sub>3</sub>	346	91	48	139 - 394
Saturation Index (100 °F) pHs		6.8	7.8	-	7.5 - 6.7
Langelier Index (100 °F)		+1.45	0	-	+0.4 - +1.4
Ryznar Index (100 °F)		5.35	7.8	-	7.1 - 5.3

TABLE 3

Concentration of Nutrients, Heavy and Trace Metals on Completion of Reservoir Filling

<u>METALS</u>		East Poplar River At Border		Increment From Soil Leaching	Range Expected On Completion Of Fill		
		1973	1974				
Iron	ppm Fe	0.26	0.88	2.95	3.21	-	3.83
Manganese	ppm Ma	0.05	0.12	0.22	0.27	-	0.34
Copper	ppm Cu	0.01	0.01	0.03	0.04		
Zinc	ppm Zn	0.025	0.02	0.07	-0.09	-	0.095
Lead	ppm Pb	L0.02	L0.02	-	L0.02		
Mercury	ppb Hg	L0.05	L0.05	0.62	0.62		
<u>NUTRIENTS</u>							
Soluble Phosphate	ppm P	0.025	0.065	0.04	0.065	-	0.105
Total Phosphate	ppm P	0.10	0.28	0.33	0.43	-	0.61
Nitrate	ppm N	0.09	0.20	0.16	0.25	-	0.36
Ammonia	ppm N	0.3	1.3	1.7	2.0	-	3.0
Chemical Oxygen Demand	ppm O <sub>2</sub>	15	42	30	45	-	72

## ANNEX E

### POPLAR RIVER ASH LEACHING TEST RESULTS

Annex E contains two reports on Poplar River ash leaching experiments conducted for the Plant, Mine and Reservoir Operations Committee by Dr. G.K. Pagenkopf of Montana State University. The reports are entitled:

- (1) Characterization of Poplar River Fly Ash Samples.  
Leachate tests described in this report were conducted with a distilled water leaching solution.
- (2) Poplar River Study III Static and Column Leaching Tests on Poplar River Ash.  
Leaching tests described in this report were conducted using a leaching solution of either as received or concentrated water from Cookson Reservoir.

These reports were re-typed to reduce space.



CHARACTERIZATION OF POPLAR RIVER FLY ASH SAMPLES

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 Montana State University  
 Bozeman, Montana 59717  
 July 17, 1978

The data presented in this report summarize the results of leaching studies conducted on Poplar River upper ash (UA), dust collector ash (DA), bottom ash (BA) and a prepared composite ash from the 1975 test burn at the Boundary Dam Generating Station.

Three ash samples were provided by the Plant, Mine and Reservoir Operations Committee. A composite ash sample was prepared by mixing 23.5% bottom ash, 4.7% upper ash and 71.8% dust collector ash. The ash samples were leached by mixing 1.0 gr of the ash with 1.0 L of water. These samples were permitted to equilibrate with atmospheric carbon dioxide which results in reduction of the solution pH. These solutions were shaken periodically for over six days when they were filtered. The filtrate was split into three fractions, one for major component analysis, one for a dissolved organic carbon and the other for boron, the boron fraction being preserved with nitric acid. The variation of pH with time is shown in Table I. This procedure is similar to that described by R.W. Talbot, M.A. Anderson, and A.W. Andren, in "Qualitative Model of Heterogeneous Equilibria in a Fly Ash Pond", University of Wisconsin, Madison.

**TABLE I Variation in pH of Fly Ash Leachate Waters with Time**

<u>Time, hrs</u>	<u>UA-0401</u>	<u>pH DA-3010</u>	<u>BA-3010</u>	<u>Composite</u>
0	10.25	10.60	9.60	10.40
19	10.28	10.72	9.95	10.65
43	10.08	10.64	9.90	10.46
67	9.80	10.41	9.75	10.37
135	9.41	10.02	9.46	9.97
165	9.16	9.96	9.27	9.80

Analysis started at hour 166.

The major component analyses and related parameters are listed in Tables II and III.

**TABLE II Chemical Analysis of Ash Leachate Solutions**

<u>Sample No.</u>	<u>HCO<sub>3</sub><sup>-</sup></u>	<u>CO<sub>3</sub><sup>=</sup></u>	<u>SO<sub>4</sub><sup>=</sup></u>	<u>Cl<sup>-</sup></u>	<u>NO<sub>3</sub><sup>-</sup></u>	<u>Na<sup>+</sup></u>	<u>K<sup>+</sup></u>	<u>Ca<sup>++</sup></u>	<u>Mg<sup>++</sup></u>
Values are mg/g Ash									
UA-0401	42.31	3.44	5.49	<1.00	<1.00	.349	.204	12.33	1.54
DA-3010	40.16	18.95	6.49	<1.00	<1.00	.602	.077	20.47	1.72
BA-3010	25.49	2.19	2.50	<1.00	<1.00	.192	1.27	7.11	1.68
Composite ash	36.42	15.22	5.79	<1.00	<1.00	.508	.164	17.45	1.56

TABLE III Measured Parameters for Ash Leachates

Sample No.	pH	S.C. mmhos/cm	$\Sigma$ cation meq/l	$\Sigma$ anion meq/l	TDS mg/gr Soil
1.0 gr Ash + 1000 gr H <sub>2</sub> O, equilibration with atmosphere					
UA-0401	9.05	.072	0.76	0.88	67.60
DA-3010	9.78	.109	1.19	1.17	90.50
BA-3010	8.98	.051	0.51	0.51	41.40
Composite Ash	9.72	.098	1.03	1.02	79.10

Trace Component Leaching

The fly ash samples were leached with diethylenetriamine pentacetic acid to obtain available trace metals. The procedure was similar to that developed by Lindsay and Novell, Agron. Abstr. p. 84, 1969. Ten grams of ash was mixed with 40 mL of DTPA solution (0.01 M CaCl<sub>2</sub>, 0.1 M triethanol amine, and 0.005 M DTPA), shaken for 2.0 hours and analyzed by atomic absorption. The concentrations of available Cd, Cu, Fe, Mn, Ni, Pb and Zn are summarized in Table IV.

A portion of the obtained extract solution was preserved for mercury analysis by adding sufficient HNO<sub>3</sub> and Cr<sub>2</sub>O<sub>7</sub><sup>2-</sup> to provide 5% (V/V) HNO<sub>3</sub> and 0.05% Cr<sub>2</sub>O<sub>7</sub><sup>2-</sup> (C. Feldman, Anal. Chem., 46, 99(1974).\*)

These samples along with a portion of the major component leachate were sent to Dr. Crawczyk, U.S. Environmental Protection Agency, Corvallis, Oregon, for boron and mercury analysis.

TABLE IV Trace Metals Extracted by DTPA From Poplar Fly Ash,  
Values in  $\mu\text{g/g}$  of Air Dry Ash

Sample No.	Cd	Cu	Fe	Mn	Ni	Pb	Zn	B
UA-0401	0.244	0.52	16.75	5.00	0.45	1.08	0.31	264
BA-3010	0.084	1.12	172.96	4.16	0.45	<1.00	0.57	586
DA-3010	0.136	0.56	7.16	2.72	1.02	1.32	0.05	215
Composite Ash	0.132	0.60	8.12	3.36	1.02	1.56	0.31	504

Leachable Carbon

A portion of the major component leachate solution was analyzed for total dissolved organic carbon using an oceanography total carbon system. The values are presented in Table V. The phenolic content was determined using a procedure developed by H. deHann. (H. deHaan, Proc. Int. Meet. Humic Sub., pp. 53-62, 1972, Centre for Agricultural Publishing and Documentation, Wageningen, The Netherlands, 1975). The procedure involved mixing 2.0 mL of water sample with 7 mL of 1% Na<sub>2</sub>CO<sub>3</sub> in 0.1 N NaOH, and 0.5 mL of Folin and Ciocalteau's phenol reagent-2 normal. The colour is developed for 30 minutes at 21° and monitored at 720 nm. The working range extends from 1-100 mg/l tyrosine using a 1 cm cell. An absorbance of 0.15 is equivalent to 10 mg/L of tyrosine. The values are listed in Table V.

TABLE V LEACHABLE CARBON

<u>Sample No.</u>	<u>DOC mg/L</u>	<u>Abs</u>	<u><math>10^3</math> Abs/DOC</u>	<u>mgC/gr Ash</u>
UA-0401	2.2	.003	1.62	2.2
DA-3010	2.4	.003	1.44	2.4
BA-3010	2.6	.004	1.51	2.6
Composite Ash	2.4	.005	2.12	2.4

The amount of readily leachable salts available from the ash samples is presented in Table VI.

TABLE VI Kilograms of Readily Leachable Salts per Cubic Meter

<u>Sample No.</u>	<u>Kg/m<sup>3</sup></u>
UA-0401	101.4
DA-3010	135.7
BA-3010	62.1
Composite Ash	118.6

The trace element analyses of the water leachate solutions are listed in Table VII.

TABLE VII Trace Metals Water Leached from Fly Ash Samples

Values are  $\mu\text{g/g}$

<u>Sample No.</u>	<u>Cd</u>	<u>Cr</u>	<u>Cu</u>	<u>Fe</u>	<u>Mn</u>	<u>Ni</u>	<u>Pb</u>	<u>Zn</u>	<u>B</u>
UA-0401	< 5.1	< 5.1	7.17	183.4	< 5.1	< 41	< 41	< 5.1	264
DA-3010	< 5.1	< 5.1	< 5.10	29.7	< 5.1	< 41	< 41	< 5.1	586
BA-3010	< 5.1	< 5.1	< 5.10	32.8	< 5.1	< 41	< 41	7.2	215
Composite Ash	< 5.1	< 5.1	< 5.10	24.6	< 5.1	< 41	< 41	5.1	504

<u>Sample No.</u>	<u>As</u>	<u>Mo</u>	<u>Sr</u>	<u>Se</u>	<u>Zr</u>	<u>Sb</u>	<u>Co</u>	<u>Al</u>	<u>Y</u>
UA-0401	< 41	< 20.5	177.2	< 41	< 5.1	< 41	< 41	-	6.15
DA-3010	< 41	< 20.5	339.1	< 41	< 5.1	< 41	< 41	-	7.17
BA-3010	< 41	< 20.5	121.9	< 41	< 5.1	< 41	< 41	788.2	< 5.10
Composite Ash	< 41	< 20.5	304.3	< 41	< 5.1	< 41	< 41	-	9.22

POPLAR RIVER STUDY III  
STATIC AND COLUMN LEACHING TESTS ON POPLAR RIVER ASH

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August 28, 1978

The studies included static and column leaching of fly ash with reservoir water and concentrated reservoir water. In addition, heated fly ash was leached with reservoir water.

The fly ash was composited according to directions; 23.5% bottom ash, 4.7% upper ash, and 71.8% dust collector ash. The static leaching test involved the mixing of 10.0 gr composite with 167 ml of the respective water (600 g/L). The heated ash composite was prepared from bottom ash that had been heated to 2000°F for 1/2 hour, and upper and dust ash that had been heated to 350°F for one hour. The concentrated reservoir water was prepared by slow evaporation of the reservoir water as received. The analyses of the leaching waters are listed in Tables 1 and 3.

Static Leaching Studies

The general procedure for the static leach tests involved mixing ash and water in 1 liter polypropylene beakers. These provide large surface area open to the atmosphere. The suspensions were shaken periodically. After a specified period the leachate water was removed and new leachate water was mixed with the ash. The leachate water was separated by decantation and filtration.

The designation of the static leaching experiment is as follows:

- FA 1, Composite + reservoir water, 10 days of contact
- FA 2, Composite + concentrated reservoir water, 10 days of contact
- FA 3, Heated composite + reservoir water, 10 days of contact
  
- FA 11, second leach, composite + reservoir water, 7 days
- FA 12, second leach, composite + concentrated reservoir water, 7 days
- FA 13, second leach, heated composite + reservoir water, 7 days
  
- FA 21, third leach, composite + reservoir water, 7 days
- FA 22, third leach, composite + concentrated reservoir water, 7 days
- FA 23, third leach, heated composite + reservoir water, 7 days
  
- FA 31, fourth leach, composite + reservoir water, 7 days
- FA 32, fourth leach, composite + concentrated reservoir water, 7 days
- FA 33, fourth leach, heated composite + reservoir water, 7 days

The analytical data is listed in Tables 1 through 4. Tables 1 and 3 quote the value in mg/L and Tables 2 and 4 present the values in mg/g ash. Particular attention was paid to the boron analysis. The procedure used was the Curcumin method and the standard curve for six different days is shown in Figure 1. The colour development was at 75°C for 20 minutes after reaching initial dryness. The composite correlation factor is 0.998.

### Column Leaching Studies

Two column leaching tests were completed using the composite ash and the two leachate waters. The columns were pyrex, 1 cm diameter, with glass frit plugs. The plug had the smallest porosity possible and when 20 grams of fly ash composite was loaded a flow of approximately 0.4 ml/hr was observed; specifically 0.43 ml/hr for the reservoir water column and 0.37 ml/hr for the concentrated water column. The specific conductance of the leachate water was monitored and the decrease with time is shown in Figure 2. The leachate water fractions were analyzed when the S.C. had decreased to 0.5 of the initial value and after the plateau had been reached. The results are listed in Tables 5 through 8. The leaching of the glass columns with 20 ml of distilled water for twenty-four hours provided a non-detectable amount of boron.

### Summary

Evaporation of the reservoir water resulted in a proportional increase in the concentrations of most components except inorganic carbon and calcium which presumably are lost through the precipitation of calcite.

The static leaching tests FA 1 and FA 2 indicate the presence of large amounts of boron. Not as much is solubilized from the heated fly ash composite but the values are much greater than those found in the original leachate water. Successive leaching indicates that most of the soluble boron is removed during the first few extractions. The same general behaviour is observed for sulfate.

The column leaching waters do not have the opportunity to equilibrate with as much carbon dioxide and thus the pH of the discharged leachate waters is higher than that observed for comparable static leachings. Sodium sulfate is removed first. Calcium and magnesium initially present in the leach waters precipitate probably as the hydroxide or carbonate salts. Boron values rise with time and amount of leachate water passed through the column. This is probably associated with a decrease in pH which increases the protonation of the boron species and thus increases the solubility. The trace metal values are all low as would be expected in solutions with such high pH values.

TABLE 1. Static Leaching of Fly Ash  
Major Components Values mg/L

<u>Sample</u>	<u>pH</u>	<u>Specific Conductance <math>\mu\text{mhos/cm}</math></u>	<u><math>\text{HCO}_3^-</math></u>	<u><math>\text{CO}_3^{2-}</math></u>	<u><math>\text{SO}_4^{2-}</math></u>	<u><math>\text{Cl}^-</math></u>	<u><math>\text{NO}_3^-</math></u>	<u><math>\text{Na}^+</math></u>	<u><math>\text{K}^+</math></u>	<u><math>\text{Ca}^{2+}</math></u>	<u><math>\text{Mg}^{2+}</math></u>	<u><math>\text{SiO}_2</math></u>
1 Std.	8.30	986	405.8	--	189	6.02	3.19	140	9.8	33.0	33.9	2.68
1 Conc.	8.88	1613	541.9	42.5	360	12.6	7.11	257	17.9	18.0	59.6	4.61
FA 1	9.01	1270	171.6	56.1	425.1	9.26	4.46	183	10.9	33.4	41.4	6.7
FA 2	9.08	1890	280.4	95.2	600.7	15.2	9.36	322.6	19.1	21.2	56.7	13.8
FA 3	8.85	1090	202.6	29.7	337.6	12.0	2.69	159.8	10.7	23.1	33.0	6.62
FA11	9.10	1140	332.0	71.3	237.5	8.82	3.17	158.2	17.7	19.0	59.2	5.79
FA12	9.22	1849	499.3	125.6	437.4	14.7	6.90	299.2	34.4	15.4	88.1	7.65
FA13	9.00	1139	384.4	51.8	229.2	8.16	2.99	160.9	17.3	17.0	60.2	5.07
FA21	9.12	1125	367.3	55.4	243.4	9.01	3.45	170.1	20.0	20.3	53.1	5.56
FA22	9.22	1805	525.8	105.5	418.1	15.2	5.84	314.3	38.4	15.6	75.2	6.88
FA23	9.03	1134	393.0	47.4	244.6	8.21	3.36	164.9	26.5	17.4	55.1	5.23
FA31	8.97	1108	381.1	47.8	222.1	5.63	2.15	166.6	20.2	20.8	52.3	4.21
FA32	9.19	1839	538.3	97.7	436.8	15.2	3.34	319.8	39.0	14.0	73.0	5.02
FA33	9.02	1086	370.5	41.3	229.6	6.94	2.34	168.6	21.1	20.6	47.3	3.74

TABLE 1. Static Leaching of Fly Ash (Continued)

<u>Sample</u>	<u><math>\Sigma</math>Cations meq/L</u>	<u><math>\Sigma</math>Anions meq/L</u>
1 Std.	10.78	10.81
1 Conc.	17.46	18.26
FA 1	13.31	13.87
FA 2	20.24	20.85
FA 3	11.08	11.72
FA 11	13.15	13.07
FA 12	21.90	22.00
FA 13	13.24	13.09
FA 21	13.28	13.25
FA 22	21.61	21.37
FA 23	12.82	13.40
FA 31	13.11	12.66
FA 32	21.61	21.66
FA 33	12.80	12.47

TABLE 2. Static Leaching of Fly Ash  
Major Components Values mg/g Ash

<u>Sample</u>	<u>pH</u>	<u>Specific Conductance</u> <u>µhos/cm</u>	<u>HCO<sub>3</sub><sup>-</sup></u>	<u>CO<sub>3</sub><sup>2-</sup></u>	<u>SO<sub>4</sub><sup>2-</sup></u>	<u>Cl<sup>-</sup></u>	<u>NO<sub>3</sub><sup>-</sup></u>	<u>Na<sup>+</sup></u>	<u>K<sup>+</sup></u>	<u>Ca<sup>2+</sup></u>	<u>Mg<sup>2+</sup></u>	<u>SiO<sub>2</sub></u>
FA 1	9.01	1270	2.87	0.94	7.10	0.15	0.07	3.06	0.18	0.58	0.69	0.11
FA 2	9.08	1890	4.68	1.59	10.03	0.25	0.16	5.39	0.32	0.35	0.95	0.23
FA 3	8.85	1090	3.38	0.50	5.64	0.20	0.04	2.67	0.18	0.39	0.55	0.11
FA11	9.10	1140	5.54	1.19	3.97	0.15	0.05	2.64	0.29	0.32	0.99	0.10
FA12	0.22	1849	8.34	2.10	7.30	0.24	0.12	5.00	0.57	0.26	1.47	0.13
FA13	9.00	1139	6.42	0.86	3.83	0.14	0.05	2.69	0.29	0.28	1.01	0.08
FA21	9.12	1125	6.13	0.92	4.06	0.15	0.06	2.84	0.33	0.34	0.89	0.09
FA22	9.22	1805	8.78	1.76	5.98	0.25	0.10	5.25	0.64	0.26	1.26	0.11
FA23	9.03	1134	6.56	0.79	4.08	0.14	0.06	2.75	0.44	0.29	0.92	0.09
FA31	8.97	1108	6.36	0.80	3.71	0.09	0.04	2.78	0.34	0.35	0.87	0.07
FA32	9.19	1839	8.99	1.63	7.29	0.25	0.06	5.34	0.65	0.23	1.22	0.08
FA33	9.02	1086	6.19	0.70	3.83	0.12	0.04	2.82	0.35	0.34	0.79	0.06

**TABLE 3. Static Leaching of Fly Ash  
Trace Components Values µg/L**

<u>Sample</u>	<u>A1</u>	<u>B</u>	<u>Cd</u>	<u>Cr</u>	<u>Cu</u>	<u>Fe</u>	<u>Mn</u>	<u>Pb</u>	<u>Zn</u>
1 Std.	<2000	820	<20	<100	<400	<350	<25	<5	<20
1 Conc.	"	1560	"	"	"	"	"	"	"
FA 1	"	16340	"	"	"	"	"	"	"
FA 2	"	17220	"	"	"	"	"	"	"
FA 3	"	10130	"	"	"	"	"	"	"
FA11	"	4780	"	"	"	"	"	"	"
FA12	"	5580	"	"	"	"	"	"	"
FA13	"	2980	"	"	"	"	"	"	"
FA21	"	2610	"	"	"	"	"	"	"
FA22	"	3300	"	"	"	"	"	"	"
FA23	"	1930	"	"	"	"	"	"	"
FA31	"	1920	"	"	"	"	"	"	"
FA32	"	2640	"	"	"	"	"	"	"
FA33	"	1420	"	"	"	"	"	"	"

TABLE 4. Static Leaching of Fly Ash  
Trace Components Values  $\mu\text{g/g}$  Ash

<u>Sample</u>	<u>Al</u>	<u>B</u>	<u>Cd</u>	<u>Cr</u>	<u>Cu</u>	<u>Fe</u>	<u>Mn</u>	<u>Pb</u>	<u>Zn</u>
FA 1	<33.4	272.9	<.33	<1.67	<.67	<5.8	<.42	<.08	<.33
FA 2	"	287.6	"	"	"	"	"	"	"
FA 3	"	169.2	"	"	"	"	"	"	"
FA11	"	79.8	"	"	"	"	"	"	"
FA12	"	93.1	"	"	"	"	"	"	"
FA13	"	49.8	"	"	"	"	"	"	"
FA21	"	43.6	"	"	"	"	"	"	"
FA22	"	55.0	"	"	"	"	"	"	"
FA23	"	32.2	"	"	"	"	"	"	"
FA31	"	32.1	"	"	"	"	"	"	"
FA32	"	44.1	"	"	"	"	"	"	"
FA33	"	23.7	"	"	"	"	"	"	"

TABLE 5. Column Leaching, Major Component Analysis  
Values - mg/L

<u>Sample</u>	<u>pH</u>	<u>Specific Conductance</u> <u>μmhos/cm</u>	<u>OH<sup>-</sup></u>	<u>CO<sub>3</sub><sup>2-</sup></u>	<u>HCO<sub>3</sub><sup>-</sup></u>	<u>SO<sub>4</sub><sup>2-</sup></u>	<u>Cl<sup>-</sup></u>	<u>NO<sub>3</sub><sup>-</sup></u>	<u>Na<sup>+</sup></u>	<u>K<sup>+</sup></u>	<u>Ca<sup>2+</sup></u>	<u>Mg<sup>2+</sup></u>	<u>SiO<sub>2</sub></u>
R.W.	8.57	994.5	--	14.8	381.1	210.0	6.10	0.72	150.0	18.6	34.6	32.6	2.42
R.W.-1	11.68	3240.0	66.9	202.7	--	639.0	33.8	10.7	507.6	45.6	15.3	0.04	12.6
R.W.-2	10.05	725.4	--	131.7	113.2	62.9	9.09	4.71	127.8	18.6	0.89	0.013	16.1
C.R.W.	8.87	1653.1	--	58.7	505.0	420.0	11.9	7.29	299.1	36.4	13.5	59.0	5.18
C.R.W.-1	11.73	4130.0	170.6	37.7	--	808.7	28.9	15.0	600.4	57.5	3.75	0.04	23.7
C.R.W.-2	10.06	1370.0	--	226.8	236.2	126.8	16.7	9.95	258.1	33.9	0.92	0.004	18.1

TABLE 5. Column Leaching (Continued)

<u>Sample</u>	<u>ΣCations</u> <u>meq/L</u>	<u>ΣAnions</u> <u>meq/L</u>
R.W.	11.34	11.30
R.W.-1	24.02	25.13
R.W.-2	6.07	7.89
C.R.W.	19.47	19.45
C.R.W.-1	27.79	29.20
C.R.W.-2	12.15	14.70

R.W. = reservoir water  
R.W.-1 = fraction collected at 20.3 ml  
R.W.-2 = fraction collected at 73.6 ml

C.R.W. = concentrated reservoir water  
C.R.W.-1 = fraction collected at 18.1 ml  
C.R.W.-2 = fraction collected at 55.6 ml

20.0 gr of fly ash

TABLE 6. Column Leaching, Major Component Analysis  
Values  $\mu\text{g/L}$

<u>Sample</u>	<u>Al</u>	<u>B</u>	<u>Cd</u>	<u>Cr</u>	<u>Cu</u>	<u>Fe</u>	<u>Mn</u>	<u>Pb</u>	<u>Zn</u>
R.W.	<2000	865.0	<20.0	<100.0	<40.0	<350.0	<25.0	<5.0	<20.0
R.W.-1	"	3206.0	"	"	"	"	"	"	"
R.W.-2	"	5348.0	"	"	"	"	"	"	"
C.R.W.	"	1745.0	"	"	"	"	"	"	"
C.R.W.-1	"	2400.0	"	"	"	"	"	"	"
C.R.W.-2	"	6165.0	"	"	"	"	"	"	"

TABLE 7. Column Leaching, Major Component Analysis  
Values - mg/g of Ash

<u>Sample</u>	<u>pH</u>	<u>Specific Conductance µhos/cm</u>	<u>OH<sup>-</sup></u>	<u>CO<sub>3</sub><sup>2-</sup></u>	<u>HCO<sub>3</sub><sup>-</sup></u>	<u>SO<sub>4</sub><sup>2-</sup></u>	<u>Cl<sup>-</sup></u>	<u>NO<sub>3</sub><sup>-</sup></u>	<u>Na<sup>+</sup></u>	<u>K<sup>+</sup></u>	<u>Ca<sup>2+</sup></u>	<u>Mg<sup>2+</sup></u>	<u>SiO<sub>2</sub></u>
R.W.-1	11.68	3240.0	0.08	0.24	--	0.65	0.034	0.01	0.52	0.05	0.016	4.1 X 10 <sup>-5</sup>	0.013
R.W.-2	10.05	725.4	--	0.48	0.42	0.23	0.025	0.02	0.47	0.07	0.003	4.8 X 10 <sup>-5</sup>	0.059
C.R.W.-1	11.73	4130.0	0.15	0.03	--	0.73	0.026	0.01	0.54	0.05	0.003	3.6 X 10 <sup>-5</sup>	0.022
C.R.W.-2	10.06	1370.0	--	0.63	0.66	0.35	0.046	0.03	0.72	0.09	0.003	1.1 X 10 <sup>-5</sup>	0.050

TABLE 8. Column Leaching, Trace Component Analysis  
Values - µg/g of Ash

<u>Sample</u>	<u>Al</u>	<u>B</u>	<u>Cd</u>	<u>Cr</u>	<u>Cu</u>	<u>Fe</u>	<u>Mn</u>	<u>Pb</u>	<u>Zn</u>
R.W.-1	<2.03	3.25	<.02	<.10	<.05	<.36	<.03	<.005	<.02
R.W.-2	<7.36	19.65	<.07	<.37	<.15	<1.29	<.09	<.018	<.07
C.R.W.-1	<1.81	2.18	<.02	<.09	<.04	<.32	<.02	<.008	<.02
C.R.W.-2	<5.56	17.13	<.06	<.28	<.11	<.97	<.07	<.014	<.06

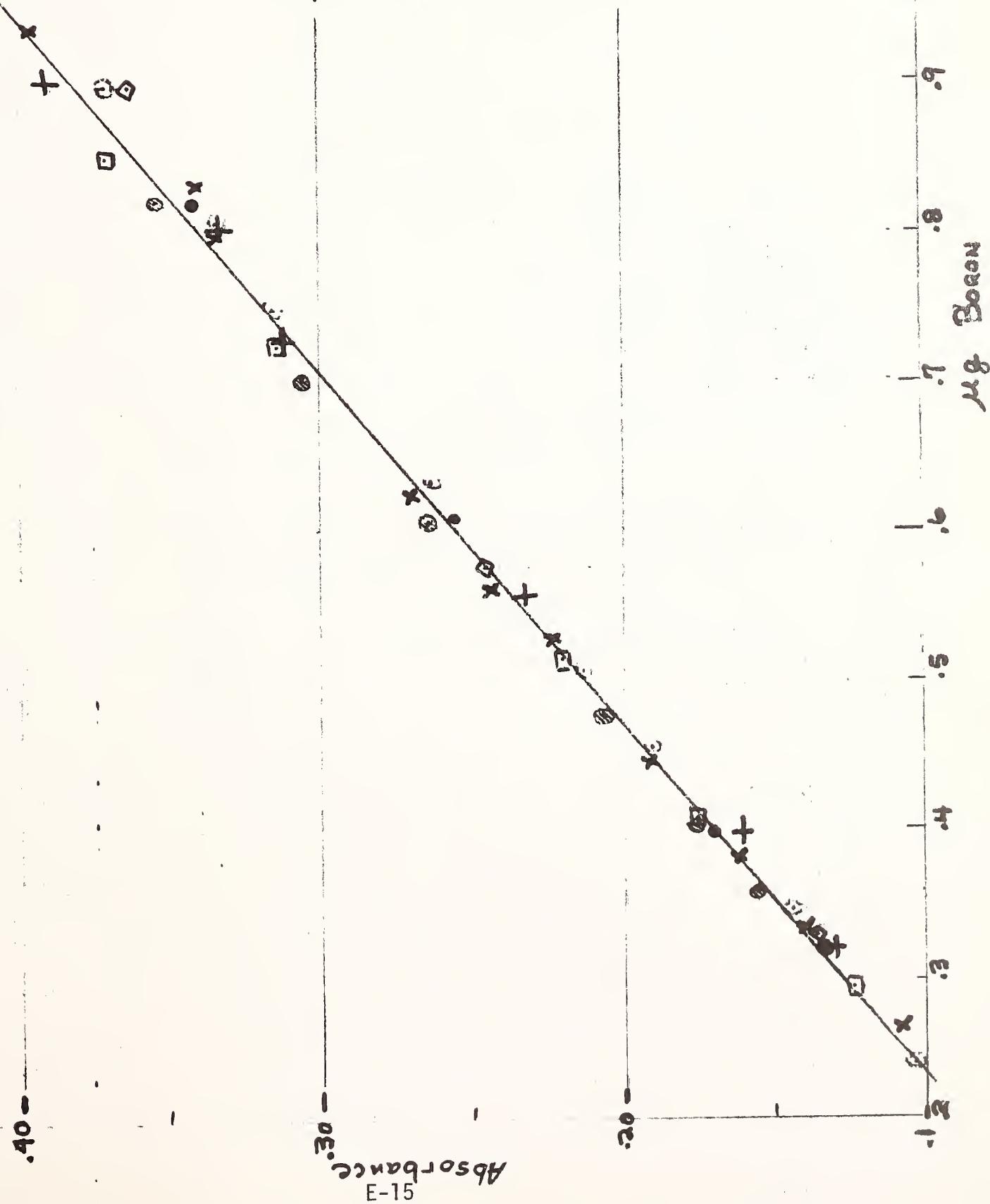
The available organic carbon is summarized in Table 9.

TABLE 9. Available Organic Carbon

<u>Sample</u>	<u>TOC, mg/g</u>	<u>Sample</u>	<u>TOC, mg/g</u>
1 Std	12.4	FA 12	--
1 Conc	24.7	FA 13	1.5
FA 1	1.1	FA 21	2.7
FA 2	1.6	FA 22	1.4
FA 3	1.3	FA 23	1.2
FA 11	1.5		

The phenolic content was not significantly different from that observed for doubly distilled water.

Figure 1. Standard Curve  
for Boron Analysis, Curcumin  
Method, Abs for 1 cm cell at  
547 nm, Data for six  
separate analyses





## ANNEX F

### BOUNDARY DAM ASH LAGOON STUDY RESULTS

Annex F contains the following:

- (1) Results of Analysis of Boundary Dam Unit #6  
Coal and Ash Samples
- (2) Chemical Report on Poplar River - Boundary Dam  
Project by G.K. Pagenkopf, Montana State  
University, February, 1979  
(This report has been re-typed to reduce space)
- (3) Boundary Dam G.S. Unit #6 Ash Lagoon Study, Ash  
Leachate Test Results, Environment Canada,  
February, 1979.



Environmental Research Laboratory  
UNITED STATES ENVIRONMENTAL PROTECTION AGENCY  
Athens, Georgia 30605

DATE: December 26, 1978

SUBJECT: INA Analysis of Solid Samples from  
the Boundary Dam Power Station

FROM: Thomas B. Hoover *Tom Hoover*  
Analytical Chemistry Branch

TO: Gary Glass  
Environmental Research Lab.  
Duluth, MN 55804

THRU: Charles H. Anderson, Chief *Charles H. Anderson*  
Analytical Chemistry Branch

Attached are Vernon Moore's results for the analysis of three samples submitted by Lee R. Rice of the Bureau of Mines. The NBS Standard Reference Material "Coal Fly Ash" was run with the samples as a control and gave results in good agreement with established values for that standard.

SSMS analyses of these samples will be reported when available.

Attachments

## BOUNDARY DAM GENERATING STATION

Sample	BDA-1	Fly Ash*	December 20, 1978
Al	$9.4 \times 10^{-2}$	Na	$2.0 \times 10^{-2}$
As	$1.4 \times 10^{-5}$	Nd	$2.4 \times 10^{-4}$
Ba	$9.3 \times 10^{-3}$	Rb	$1.9 \times 10^{-5}$
Br	$2.7 \times 10^{-6}$	Sb	$4.1 \times 10^{-6}$
Ca	$8.1 \times 10^{-2}$	Sc	$1.4 \times 10^{-5}$
Ce	$1.1 \times 10^{-4}$	Se	$9.0 \times 10^{-6}$
Co	$6.1 \times 10^{-6}$	Sm	$7.4 \times 10^{-6}$
Cr	$3.7 \times 10^{-5}$	Sr	$4.5 \times 10^{-3}$
Cs	$2.6 \times 10^{-6}$	Ta	$2.1 \times 10^{-6}$
Dy	$8.3 \times 10^{-6}$	Tb	$1.4 \times 10^{-6}$
Eu	$1.1 \times 10^{-6}$	Te	$8.8 \times 10^{-5}$
Fe	$2.2 \times 10^{-2}$	Th	$2.7 \times 10^{-5}$
Hf	$1.3 \times 10^{-5}$	Ti	$4.8 \times 10^{-3}$
La	$5.7 \times 10^{-5}$	U	$1.1 \times 10^{-5}$
Lu	$6.2 \times 10^{-7}$	V	$6.4 \times 10^{-5}$
Mg	$2.5 \times 10^{-2}$	W	$4.5 \times 10^{-6}$
Mn	$2.1 \times 10^{-4}$	Yb	$3.5 \times 10^{-6}$
Mo	$3.0 \times 10^{-5}$	Zr	$3.3 \times 10^{-5}$
			$5.1 \times 10^{-4}$

\*All values: g element/g sample

## BOUNDARY DAM GENERATING STATION

Sample	BDL-1	Lignite*	December 20, 1978
Al	$1.4 \times 10^{-2}$	Na	$3.9 \times 10^{-3}$
As	$2.1 \times 10^{-6}$	Nd	$4.8 \times 10^{-5}$
Ba	$1.2 \times 10^{-3}$	Rb	$3.3 \times 10^{-6}$
Br	$1.2 \times 10^{-6}$	Sb	$5.2 \times 10^{-7}$
Ca	$1.1 \times 10^{-2}$	Sc	$2.3 \times 10^{-6}$
Ce	$1.8 \times 10^{-5}$	Se	$2.5 \times 10^{-6}$
Co	$1.0 \times 10^{-6}$	Sm	$1.2 \times 10^{-6}$
Cr	$5.5 \times 10^{-6}$	Sr	$3.1 \times 10^{-4}$
Cs	$4.1 \times 10^{-7}$	Ta	$3.9 \times 10^{-7}$
Dy	$1.8 \times 10^{-6}$	Tb	$2.3 \times 10^{-7}$
Eu	$1.5 \times 10^{-7}$	Te	$1.1 \times 10^{-5}$
Fe	$4.4 \times 10^{-3}$	Th	$4.0 \times 10^{-6}$
Ga	$1.2 \times 10^{-5}$	Ti	$7.1 \times 10^{-4}$
Hf	$1.7 \times 10^{-6}$	U	$1.5 \times 10^{-6}$
La	$8.8 \times 10^{-6}$	V	$8.6 \times 10^{-6}$
Lu	$1.1 \times 10^{-7}$	W	$2.4 \times 10^{-6}$
Mg	$3.2 \times 10^{-3}$	Yb	$6.0 \times 10^{-7}$
Mn	$2.8 \times 10^{-5}$	Zn	$1.1 \times 10^{-5}$
Mo	$2.9 \times 10^{-6}$	Zr	$6.4 \times 10^{-5}$

\*All values: g element/g sample

## BOUNDARY DAM GENERATING STATION

Sample	NBS	Coal Fly Ash*	Date
Al	$1.0 \times 10^{-1}$	Na	$2.1 \times 10^{-3}$
As	$4.1 \times 10^{-5}$	Nd	$2.7 \times 10^{-4}$
Ba	$2.1 \times 10^{-3}$	Rb	$9.6 \times 10^{-5}$
Br	$6.6 \times 10^{-6}$	Sb	$3.9 \times 10^{-6}$
Ca	$3.7 \times 10^{-2}$	Sc	$2.5 \times 10^{-5}$
Ce	$1.7 \times 10^{-4}$	Se	$5.8 \times 10^{-6}$
Co	$3.4 \times 10^{-5}$	Sm	$1.2 \times 10^{-5}$
Cr	$1.2 \times 10^{-4}$	Sr	$1.0 \times 10^{-3}$
Cs	$8.6 \times 10^{-6}$	Ta	$1.9 \times 10^{-6}$
Dy	$1.1 \times 10^{-5}$	Tb	$2.4 \times 10^{-6}$
Eu	$2.7 \times 10^{-6}$	Te	$6.6 \times 10^{-5}$
Fe	$6.5 \times 10^{-2}$	Th	$2.3 \times 10^{-5}$
Ga	$1.9 \times 10^{-4}$	Ti	$6.4 \times 10^{-3}$
Hf	$8.6 \times 10^{-6}$	U	$8.6 \times 10^{-6}$
K	$1.5 \times 10^{-2}$	V	$1.7 \times 10^{-4}$
La	$8.8 \times 10^{-5}$	W	$3.5 \times 10^{-6}$
Lu	$1.2 \times 10^{-6}$	Yb	$5.4 \times 10^{-6}$
Mg	$1.4 \times 10^{-2}$	Zn	$2.4 \times 10^{-4}$
Mn	$4.5 \times 10^{-4}$	Zr	$2.2 \times 10^{-4}$
Mo	$3.2 \times 10^{-5}$		

\* All values: g element/g sample

## BOUNDARY DAM GENERATING STATION

Sample	BDA -22	Bottom Ash*	December 20, 1978
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Al	$6.9 \times 10^{-2}$	Na	$1.7 \times 10^{-2}$
As	$3.5 \times 10^{-6}$	Nd	$1.5 \times 10^{-4}$
Ba	$7.8 \times 10^{-3}$	Rb	$2.2 \times 10^{-5}$
Br	$1.5 \times 10^{-6}$	Sb	$1.3 \times 10^{-6}$
Ca	$7.2 \times 10^{-2}$	Sc	$1.3 \times 10^{-5}$
Dy	$6.0 \times 10^{-6}$	Se	$5.1 \times 10^{-6}$
Ce	$1.0 \times 10^{-4}$	Sm	$7.9 \times 10^{-6}$
Co	$6.4 \times 10^{-6}$	Sr	$2.6 \times 10^{-3}$
Cr	$3.5 \times 10^{-5}$	Ta	$2.1 \times 10^{-6}$
Cs	$2.9 \times 10^{-6}$	Tb	$1.5 \times 10^{-6}$
Eu	$1.1 \times 10^{-6}$	Te	$7.4 \times 10^{-5}$
Fe	$3.3 \times 10^{-3}$	Th	$2.5 \times 10^{-5}$
Ga	$8.6 \times 10^{-5}$	Ti	$3.9 \times 10^{-3}$
Hf	$3.9 \times 10^{-5}$	U	$9.8 \times 10^{-6}$
La	$5.7 \times 10^{-5}$	V	$5.2 \times 10^{-5}$
Lu	$9.0 \times 10^{-7}$	W	$3.4 \times 10^{-6}$
Mg	$1.9 \times 10^{-2}$	Yb	$3.7 \times 10^{-6}$
Mn	$1.4 \times 10^{-4}$	Zn	$1.2 \times 10^{-4}$
Mo	$2.0 \times 10^{-5}$	Zr	$4.6 \times 10^{-4}$

\*All values: g element/g sample

CHEMICAL REPORT  
Poplar River - Boundary Dam Project

IV

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February, 1979

<u>CONTENTS</u>	<u>PAGE</u>
Field Waters.....	F-7
Laboratory Studies.....	F-13
Poplar River Studies.....	F-14

<u>TABLES</u>	
Major Component Analyses.....	F-8
Trace Component Analyses.....	F-10
Percent Solids in Slurries.....	F-12
Organic Carbon.....	F-13
Boron Leaching.....	F-16

<u>FIGURES</u>	
Boron Leaching with Time.....	F-17

<u>APPENDIX</u>	
Analytical Procedures.....	F-18
Solid-Solution Separation.....	F-18
Supplemental Analyses.....	F-18
Sample Descriptions.....	F-19

This report includes chemical analysis data for a group of water samples collected by the Boundary Dam power plant operation, the results of leaching studies using Boundary Dam power plant fly ash, and water quality predictions for the Poplar River Project.

### Field Waters

The major component analyses are listed in Table 1 and the trace component analyses are listed in Table 2. A summary of the procedures used to analyze the samples is included in the appendix.

The water samples BDW-6, BDW-8, BDW-9, and BDW-10 are slurry waters collected August 17, 18, 21, 22 and 23, 1978 respectively. Sample SW is a composite of slurry waters collected at the same time from #6 ash sluice pump. Chemical analyses for these six samples are very similar. The major anions are sulfate and bicarbonate with the cations being dominated by sodium, calcium and magnesium. The calculated dissolved solids are approximately 600 mg/l with nearly fifty percent being due to sulfate and sodium. The concentrations of trace elements are within the ranges considered to represent background except for zinc in BDW-9, 154 µg/l. This may represent contamination.

Water samples BDW-1, BDW-2, BDW-3, BDW-4, and BDW-5 were collected from the ash lagoon overflow on August 17, 18, 21, 22, and 23, 1978 respectively. As with the preceding group, they exhibit homogeneity within the group. However, they are considerably different from the first group. Among the major constituents, the concentrations of sulfate, sodium and calcium exhibit marked increases. Total dissolved solids exhibit a 50% increase to near 900 mg/l. The pH increases some from the low to mid eight range to the mid to high eight range. The concentrations of sodium and sulfate increase by an equivalent amount and the increase in calcium concentration is sufficient to bring the water to near saturation with respect to calcium sulfate and calcium carbonate.

There are also dramatic increases in the concentrations of some trace constituents. Most notable is boron where the concentrations increase from less than 0.4 mg/l to over 2.7 mg/l. Sizeable increases are also observed for arsenic, selenium, and fluoride. The average arsenic value increased from 3.8 to 14.6 µg/l. The average selenium value increased from less than 0.1 µg/l to 20.6 µg/l, and the average fluoride value from 0.25 to 1.93 mg/l.

Two groups of slurry samples were collected during the same time sequence. Five of these, BDS-1, BDS-2, BDS-3, BDS-4, and BDS-5, were precipitator ash or upper ash slurry and the other five, BDS-6, BDS-7, BDS-8, BDS-9, and BDS-10, came from the bottom ash slurry. There is sizeable variation in the amount of filterable solid material present in these samples as evidenced by the results listed in Table 3. One of these samples, BDS-10, was destroyed during transport. The procedure for separation of solution and solids is included in the appendix.

**Table 1. Major Component Analyses of Boundary  
Dam Water Samples. Values in mg/l**

<u>Sample</u>	<u>S.C.</u> <u>mmhos/cm</u>	<u>pH</u>	<u>HCO<sub>3</sub></u>	<u>CO<sub>3</sub></u>	<u>OH</u>	<u>SO<sub>4</sub></u>	<u>Cl</u>	<u>NO<sub>3</sub></u>
S.W.	0.740	8.25	214	--	--	210	10.3	0.93
BDW-1	1.132	8.86	169	14.7	--	449	10.4	0.71
BDW-2	1.129	8.81	169	12.7	--	458	11.9	1.04
BDW-3	1.139	8.51	193	4.3	--	453	11.4	0.56
BDW-4	1.144	8.52	192	5.0	--	437	11.8	0.58
BDW-5	1.137	8.93	168	17.4	--	433	11.4	1.07
BDW-6	0.733	8.45	206	3.7	--	208	12.0	0.43
BDW-7	0.723	8.23	194	--	--	204	9.4	0.47
BDW-8	0.744	8.25	213	--	--	208	9.4	0.73
BDW-9	0.757	8.17	215	--	--	204	9.8	1.18
BDW-10	0.749	8.54	204	4.8	--	212	9.6	1.29
FA-1	0.730	10.59	29.8	57.1	6.61	220	10.0	1.99
FA-2	0.705	10.21	51.6	41.1	2.75	210	13.6	2.04
FA-3	0.706	9.73	41.4	43.1	--	231	10.5	1.90
FA-4	0.733	9.72	36.8	55.6	--	273	13.0	2.59
BA-1	0.790	8.96	172	14.4	--	275	10.0	0.91
BA-2	0.776	9.04	174	12.0	--	245	11.5	0.89
BA-3	0.782	8.63	183	4.31	--	234	10.9	0.15
BA-4	0.784	8.86	192	10.0	--	235	10.4	0.57
CA-1	0.765	10.33	39.5	41.6	3.63	262	10.0	1.86
CA-2	0.732	10.10	81.2	50.4	2.14	235	10.8	0.80
CA-3	0.747	9.55	64.8	35.9	--	240	11.1	0.05
CA-4	0.753	9.49	94.5	36.9	--	240	11.6	0.37
BDS-1	1.379	11.47	5.0	73.3	50.2	375	9.7	0.76
BDS-2	0.726	9.07	149.9	8.6	--	231	10.1	1.47
BDS-3	2.633	12.07	2.08	120.9	198.9	81.9	11.0	1.75
BDS-4	1.600	11.50	5.13	80.2	53.7	309	11.8	1.24
BDS-5	1.654	11.67	3.34	76.9	79.5	225	13.7	1.24
BDS-6	1.032	10.44	47.7	17.9	4.68	372	10.2	0.48
BDS-7	0.821	9.84	59.5	20.3	--	280	10.1	0.52
BDS-8	0.794	10.01	50.8	25.5	1.73	287	9.2	0.41
BDS-9	0.874	8.81	100.5	3.2	--	287	11.2	10.9 ?

Cont'd.

Table 1. Major Component Analyses of Boundary  
Dam Water Samples. Values in mg/L.

Samples	Na	K	Ca	Mg	SiO <sub>2</sub>	B	Cations meq/L	Anions meq/L
S.W.	92.2	11.2	44.4	22.3	4.68	0.343	8.34	8.19
BDW-1	190.5	11.6	58.6	21.2	4.04	2.628	13.25	12.92
BDW-2	187.2	12.0	58.6	21.2	4.74	2.750	13.11	13.08
BDW-3	189.7	11.7	62.0	20.1	4.04	2.765	13.29	13.08
BDW-4	187.7	11.7	60.0	19.7	3.80	2.760	13.09	12.76
BDW-5	182.7	11.8	59.3	19.7	4.27	2.801	12.83	12.69
BDW-6	93.9	11.1	44.5	21.0	3.11	0.388	8.31	8.19
BDW-7	93.3	11.2	45.1	21.2	2.66	0.356	8.34	7.69
BDW-8	95.0	11.3	45.6	21.3	2.66	0.368	8.45	8.10
BDW-9	94.3	11.8	45.6	22.1	2.44	0.345	8.49	8.07
BDW-10	94.8	11.4	45.7	22.0	2.22	0.375	8.51	8.21
FA-1	134.7	13.4	39.9	0.31	8.83	4.315	8.09	7.67
FA-2	127.2	16.9	37.2	0.38	7.10	5.238	7.85	7.17
FA-3	114.9	12.6	38.5	0.48	5.09	5.562	7.28	7.26
FA-4	117.5	14.0	50.1	0.61	2.59	6.306	8.02	8.55
BA-1	123.6	11.7	42.9	19.6	3.64	2.023	9.44	9.43
BA-2	124.8	12.4	35.2	21.0	4.21	2.297	9.24	8.69
BA-3	108.0	12.5	32.5	23.2	5.69	2.749	8.55	8.33
BA-4	113.6	12.8	31.9	26.1	3.19	2.479	9.01	8.67
CA-1	131.0	26.4	52.4	0.51	5.94	4.636	9.03	8.02
CA-2	129.4	14.1	45.1	0.31	3.06	5.260	8.27	8.36
CA-3	112.7	12.6	49.1	0.69	3.19	5.515	7.73	7.58
CA-4	114.3	11.5	56.4	1.21	7.39	6.086	8.17	8.11
BDS-1	130.4	13.1	130.3	0.04	6.35	5.613	12.51	13.56
BDS-2	83.1	12.0	39.8	21.7	3.68	1.281	7.70	7.86
BDS-3	209.5	12.1	97.4	0.01	6.92	2.515	14.28	17.81
BDS-4	184.2	13.5	35.3	0.05	7.50	4.788	10.12	12.70
BDS-5	155.4	12.5	78.2	0.06	11.2	5.582	10.98	12.38
BDS-6	194.1	12.6	27.4	0.35	13.64	4.830	10.14	9.43
BDS-7	132.5	14.3	42.8	0.13	8.94	4.425	8.27	7.78
BDS-8	146.4	13.5	26.7	0.47	5.82	3.914	8.08	8.02
BDS-9	127.8	13.3	31.2	12.1	5.58	4.086	8.45	8.24

Table 2. Trace Component Analyses of Boundary Dam Water Samples

<u>Sample</u>	<u>Al</u> mg/l	<u>As</u> μg/l	<u>Cd</u> μg/l	<u>Cr</u> mg/l	<u>Cu</u> μg/l	<u>F</u> mg/l
BDW-1	1.85	16.89	<1.00	<0.05	<10.00	1.93
BDW-2	1.36	16.25		<0.05		1.92
BDW-3	0.87	11.40		<0.05		1.92
BDW-4	0.87	11.40		<0.05		1.95
BDW-5	1.85	16.89		<0.05		1.91
BDW-6	<0.05	1.71		<0.05		0.25
BDW-7	<0.05	2.83		<0.05		0.24
BDW-8	<0.05	4.91		<0.05		0.24
BDW-9	<0.05	4.45		<0.05		0.25
BDW-10	<0.05	4.68		<0.05		0.25
BDS-1	12.72	<1.00		0.20		2.36
BDS-2	1.11	4.68		<0.05		0.48
BDS-3	16.48	<1.00		<0.05		1.92
BDS-4	14.20	<1.00		0.11		1.31
BDS-5	7.60	<1.00		0.20		3.18
BDS-6	4.53	1.67		<0.05		0.12
BDS-7	6.69	<1.00		<0.05		0.25
BDS-8	6.35	1.27		<0.05		0.15
BDS-9	0.20	8.13		<0.05		0.38
FA-1	4.76	<1.00		0.11		1.79
FA-2	4.67	<1.00		0.11		1.56
FA-3	5.97	1.22		0.14		1.50
FA-4	7.01	<1.00		0.16		1.54
BA-1	0.68	<1.00		<0.05		0.21
BA-2	0.88	<1.00		<0.05		0.18
BA-3	0.24	<1.00		<0.05		0.15
BA-4	1.28	<1.00		<0.05		0.15
CA-1	4.96	<1.00		<0.05		1.69
CA-2	6.32	<1.00		<0.05		1.50
CA-3	5.45	1.23		<0.05		1.36
CA-4	6.07	5.54		<0.05		1.32
BA-3030	0.24	<1.00		<0.05		0.17
SW	<0.05	4.09	↓	<0.05	↓	0.24

Table 2. Continued

Sample	Fe µg/l	Mn µg/l	Pb µg/l	Se µg/l	Zn µg/l
BDW-1	<30.0	<5.0	<10.0	27.0	<5.0
BDW-2				16.8	<5.0
BDW-3				14.8	<5.0
BDW-4				23.6	<5.0
BDW-5				20.6	<5.0
BDW-6				<1.0	<5.0
BDW-7				<1.0	<5.0
BDW-8				<1.0	<5.0
BDW-9				<1.0	154.0?
BDW-10				<1.0	<5.0
BDS-1			12.00	89.0	<5.0
BDS-2			<10.00	8.4	<5.0
BDS-3			<10.00	36.6	<5.0
BDS-4			<10.00	43.0	<5.0
BDS-5			28.00	66.2	<5.0
BDS-6			12.00	<1.0	<5.0
BDS-7			<10.00	<1.0	<5.0
BDS-8				<1.0	<5.0
BDS-9				<1.0	<5.0
FA-1				72.5	<5.0
FA-2				71.3	<5.0
FA-3				60.1	<5.0
FA-4				40.0	7.0
BA-1				<1.0	<5.0
BA-2				<1.0	<5.0
BA-3				<1.0	<5.0
BA-4				<1.0	<5.0
CA-1				68.6	201.0?
CA-2				29.3	<5.0
CA-3				60.1	<5.0
CA-4				30.0	<5.0
BA-3030				<1.0	<5.0
SW				<1.0	<5.0

Table 3 Solid-Solution Composition of Slurry Samples

<u>Sample</u>	<u>Grams Solids</u>	<u>Grams Water</u>	<u>% Solids</u>
BDS-1	7.93	1049	0.75
BDS-2	5.37	511	10.4
BDS-3	109.0	908	10.7
BDS-4	54.0	992	5.16
BDS-5	64.0	919	6.51
BDS-6	71.7	929	7.16
BDS-7	90.9	754	10.8
BDS-8	31.6	931	3.28
BDS-9	370.0	844	30.5

The chemical analyses for the solutions are listed in Tables 1 and 2. Comparison of the specific conductance values with the percent filterable solids present in the slurries indicates no correlation. BDS-1 has one of the lowest % solids but and intermediate S.C. whereas BDS-9 has the highest % solids but one of the lowest S.C. values. The specific conductance provides a good indication of dissolved solids content and thus there does not appear to be a fixed percentage of soluble material in these solids. As a group, BDS-1 through BDS-5 has the highest specific conductance and pH. The pH of the solutions will go down as more carbon dioxide dissolves and neutralizes the basic species. Sulfate and sodium exhibit sizeable variation. Chloride and potassium show little variation as does nitrate except for one value (BDS-9) that probably represents contamination. Magnesium is low because of the high pH. For the one sample where the pH is comparable to the other waters, BDS-2, the magnesium concentration is also comparable. What magnesium that is leached from the ash will reprecipitate as the hydroxide and will subsequently redissolve when the pH goes down.

The average concentrations of silica and boron are higher than those observed in BDW-1 through BDW-5. However, the average increase is not a factor of 2. The high pH values increase the concentration of aluminum as would be expected through the formation of  $\text{Al}(\text{OH})_4^-$ . This concentration will decrease as the pH goes down. Iron, manganese, zinc, cadmium, and copper concentration are all low. Since these metals form soluble hydroxide species, their concentrations could be higher. A somewhat higher concentration of lead is observed in BDS-1, BDS-5, BDS-6 and the highest chromium concentrations are observed in BDS-1, BDS-4, and BDS-5. Sizeable variation is observed for arsenic with no values as high as those observed for BDW-1 through BDW-5. Selenium concentrations in BDS-1 through BDS-5 are higher than those observed in BDW-1 through BDW-5. As with some other elements, fluoride exhibits sizeable variation over the range observed for the other water samples.

The concentrations of dissolved organic carbon are listed in Table 4. The water samples had been filtered prior to total organic carbon analysis and thus the values are higher by about 2 mg/l than would

be observed for the unfiltered samples. None of the values is considered excessive especially since the solid material separated from the slurry samples contained some coal.

Table 4 Organic Carbon Analyses

<u>Sample</u>	<u>C, mg/l</u>	<u>Sample</u>	<u>C, mg/l</u>
BDW-1	10.3	SW	12.3
BDW-2	7.8	BDS-1	4.7
BDW-3	7.4	BDS-2	8.8
BDW-4	8.1	BDS-3	4.5
BDW-5	8.1	BDS-4	3.1
BDW-6	12.6	BDS-5	4.2
BDW-7	11.2	BDS-6	2.3
BDW-8	11.1	BDS-7	2.2
BDW-9	12.3	BDS-8	2.2
BDW-10	9.4	BDS-9	10.7
D.D.H <sub>2</sub> O	1.6	BA-4	9.2
filtered D.D.H <sub>2</sub> O	3.3	CA-4	8.0
		FA	4.9

#### Laboratory Studies

A sequence of leaching studies has been conducted using provided Boundary Dam fly ash and bottom ash samples. Five fly ash samples were collected August 17, 18, 21, 22, and 23, 1978. These samples were composited to form a 400 gr upper ash sample, designated as FA. During the same time frame, bottom ash was collected and composited to form a 200 gram sample. This sample is designated BA. Portions of these two samples were combined, 42% BA and 58% FA by weight, to form another sample designated CA.

These three samples were leached with slurry water SW using static conditions. Solids were mixed at a rate of 55.0 gr/l in teflon beakers that provided sample opportunity for the absorption of atmospheric carbon dioxide. The slurries were periodically stirred and aliquots (~150 ml) were removed after 10, 17, 24 and 31 days of contact. The analytical results are listed in Tables 1 and 2.

Contact of the SW water with the ash samples results in dramatic changes in some chemical parameters and minimal if any changes in others.

The specific conductance remains fairly constant indicating that total dissolved solids does not change dramatically. There is a sizable quantity of basic oxides present in the fly ash samples and as a consequence the pH of FA-1, FA-2, CA-1 and CA-2 is high. In this pH range magnesium is precipitated. A modest increase is observed in sulfate, sodium and calcium concentrations. The basic oxides are not as

prevalent in the bottom ash sample and thus the pH of the leaching solutions is lower and the magnesium is not precipitated. The presence of strong base facilitates the dissolution of aluminum silicates which will subsequently reprecipitate when the pH goes down due to the absorption of carbon dioxide.

The concentrations of the trace metals cadmium, copper, iron, manganese, lead, and zinc do not appear to be elevated by the leaching process. Fluoride and arsenic exhibit modest variability. The selenium concentration is much greater in the FA and CA sequences and indicates that the selenium in the BDW-1 through BDW-5 samples is coming from the leaching of the fly ash. Boron concentrations are also much greater in the leachate samples than in the SW water.

### Poplar River Studies

One of the objectives of these studies was to obtain sufficient data to make reasonable predictions regarding the water quality of the Poplar River project. These data, along with a variety of experiments previously completed and the results of additional experiments, will be utilized to project what may happen at the Poplar River site. The two chemical components that appear to be a major concern are sulfate and boron.

The concentration of sulfate in the Boundary Dam slurry waters and the Poplar River reservoir water are similar, approximately 200 mg/l. When these waters contact the ash, the concentration of sulfate will increase, with the degree of increase being limited by the amount of readily soluble sulfate present in the ash and the solubility of calcium sulfate. The Boundary Dam ash contains approximately 1.2 mg/g and thus if the ash is mixed at a rate of 55 g/l the sulfate concentration will increase by 75 mg/l. The amount of sulfate in the Poplar River ash samples is greater and variable: upper ash 5.5 mg/g, dust ash 6.5 mg/g, bottom ash 2.5 mg/g, and the composite 5.3 mg/g. If the composite is mixed with the reservoir water at a rate of 55 g/l, the sulfate concentration could increase by 319 to 508 mg/l. If only bottom ash is mixed, the maximum predicted concentration is 327 mg/l. Using a solubility product for calcium sulfate of  $10^{-5.2}$  and an average calcium concentration of 40 mg/l ( $1.0 \times 10^{-3}M$ ) the predicted sulfate solubility limit would be 605 mg/l. This limit does impose some restrictions on the utilization of the water.

The boron concentrations are of more concern. A variety of leaching studies have been completed and these are summarized in Table 5. Many of the studies are time dependent and these data are presented graphically in Figure 1.

Steady state concentrations of boron are approached after thirty days of contact. The Boundary Dam leaching studies approach a limiting boron concentration of 45  $\mu\text{g/g}$  for the bottom ash and 110  $\mu\text{g/g}$  for the fly and the composited ash samples. The mixing of ash and sluice water

at a rate of 55 g/l would provide boron concentrations of 2.5 mg/l and 6.1 mg/l respectively. Boron concentrations in this range are observed in the BDS and BDW-1 through BDW-5 water samples.

Using these results as a guideline and comparable leaching studies for the Poplar River composite ash provides available boron of 400  $\mu\text{g/g}$ . When the ash and reservoir water are mixed at a rate of 55 g/l a boron concentration as high as 22 mg/l can be expected. A longer study, 71 days, provided a boron concentration of 16 mg/l. These results establish the range of boron concentrations that can be expected in the settling ponds.

Table 5. Boron Leaching Studies

<u>Sample</u>	<u>µB/g</u>	<u>Comments</u>	
UA-0401	264	1 g Poplar River ash plus 1.0 liter distilled water,	
DA-3010	586		7 day contact
BA-3010	215		
Composite Ash	504		
FA01	259	Poplar River Composite ash	9 day contact
FA11	325	+ Poplar River reservoir	17 day contact
FA21	355	water, 60 g/L	24 day contact
FA31	375		31 day contact
FA02	261	Poplar River Composite Ash	9 day contact
FA12	328	and concentrated Poplar River	17 day contact
FA22	357	Reservoir water, 60 g/L	24 day contact
FA32	375		31 day contact
FA03	155	Heated Poplar River Composite	9 day contact
FA13	191	Ash + Poplar River reservoir	17 day contact
FA23	210	water 60 g/L	24 day contact
FA33	222		31 day contact
FA1	72	Boundary Dam fly ash plus	9 day contact
FA2	89	slurry water	16 day contact
FA3	95		23 day contact
FA4	108	55 g/L	30 day contact
BA1	30.6	Boundary Dam bottom ash plus	9 day contact
BA2	35.5	slurry water	16 day contact
BA3	43.8		23 day contact
BA4	38.8	55 g/L	30 day contact
CA1	78	Boundary Dam composite ash	9 day contact
CA2	89	plus slurry water	16 day contact
CA3	94		23 day contact
CA4	104	55 g/L	30 day contact
PR-Comp	294	Poplar River Composite plus	71 day contact
		Reservoir water 30 g/500 ml	
PR-BA	105	Poplar River Bottom Ash plus	14 day contact
		30 g/500 ml	

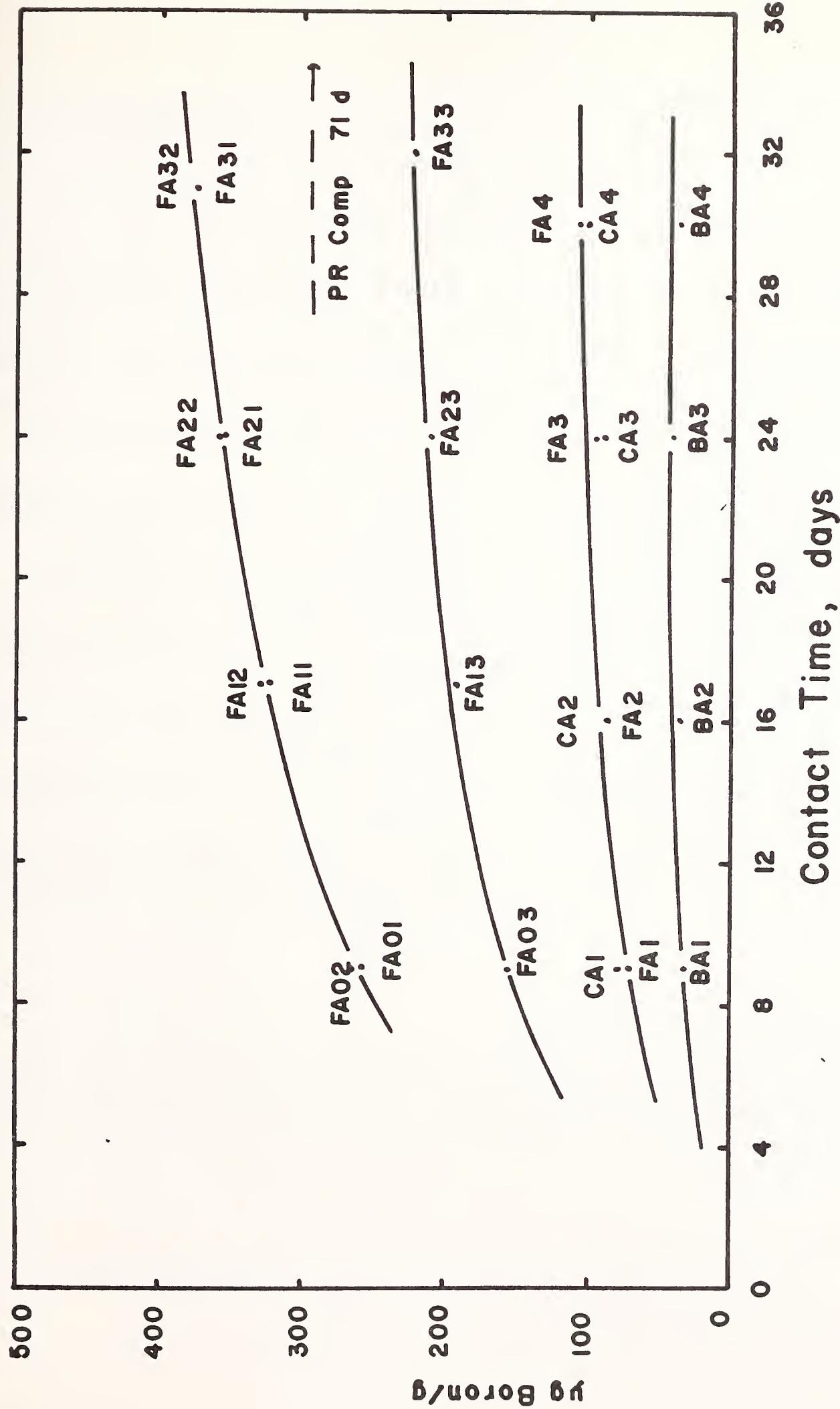


Figure 1. Time Dependence of Boron Leaching

## APPENDIX

### Analytical Procedures

Specific Conductance	Determined by standardized probe and corrected to 25°C.
pH	Measured by standardized electrodes, glass and SCE reference.
$\text{HCO}_3^-$ , $\text{CO}_3^{2-}$ , $\text{OH}^-$ .	These are determined potentiometrically (pH) by titration with standard acid.
$\text{SO}_4^{2-}$	Titration with standard barium using nitrosulfazo III indicator for equivalence point detection.
$\text{Cl}^-$	Spectrophotometric using $\text{Hg}(\text{SCN})_2$ and Iron (III).
$\text{NO}_3^-$	Brucinic sulfate and/or spectrophotometric method
Na, K, Ca, Mg	Determined by atomic absorption spectrophotometry using appropriate radiation buffers.
$\text{SiO}_2$	Spectrophotometric using molybdate-silicate procedure.
Boron	Curcumin method, spectrophotometric with color development at 75°C.
Al, Cd, Cr, Cu, Fe, Mn, Pb, Zn	Determined by atomic absorption spectrophotometry.
As and Se	Hydride generation followed by detection by atomic absorption spectrophotometry.
F	Ion selective electrode.

### Solid-Solution Separation

The suspensions were filtered through weighed Whitman 42 filter paper. The filtrate was then passed through a weighed  $0.45 \mu$  membrane filter. The two papers were dried slowly with a heat lamp and weighed. The major problem lies with loss of volatile components, however, this is probably small compared to the total solids available.

### Analyses by EPA Laboratories

Splits of the water samples were sent to Mr. Dan Krawczyk, EPA Corvallis, Oregon and Dr. Charles Anderson, EPA Athens, Georgia for chemical analysis. No results have been obtained.

## BOUNDARY DAM POWER STATION SAMPLES

- BDA-3      Composite 400-gram sample of upper ash prepared by mixing equal weights of five separate dry fly ash samples.
- BDA-17     Approximately 200 grams of fly ash sample collected on August 17, 1978.
- BDA-18     Approximately 200 grams of fly ash sample collected on August 18, 1978.
- BDA-19     Approximately 200 grams of fly ash sample collected on August 19, 1978.
- BDA-20     Approximately 200 grams of fly ash sample collected on August 20, 1978.
- BDA-21     Approximately 200 grams of fly ash sample collected on August 21, 1978.

NOTE: All bottom ash samples were dried at 105°C to constant weight and were ground in a Bico pulverizer with ceramic plates (some alumina contamination of these samples will have resulted from this grinding process).

- BDA-37    Composite 200-gram sample of bottom ash prepared by mixing equal weights of five dry pulverized bottom ash samples.
- BDA-38    Approximately 200 grams of dried, pulverized bottom ash sample collected on August 17, 1978.
- BDA-39    Approximately 200 grams of dried, pulverized bottom ash sample collected on August 18, 1978.
- BDA-40    Approximately 200 grams of dried, pulverized bottom ash sample collected on August 19, 1978.
- BDA-41    Approximately 200 grams of dried, pulverized bottom ash sample collected on August 20, 1978.
- BDA-42    Approximately 200 grams of dried, pulverized bottom ash sample collected on August 21, 1978.

- BDL-3      Composite 200-gram sample of lignite prepared by mixing equal weights of five dried, pulverized lignite samples.
- BDS-1      Sample of "precipitator ash" (upper ash slurry) collected at end of ash line on August 17, 1978.
- BDS-2      Sample of upper ash slurry collected at end of ash line on August 18, 1978.
- BDS-3      Sample of upper ash slurry collected at end of ash line on August 21, 1978.
- BDS-4      Sample of upper ash slurry collected at end of ash line on August 22, 1978 (sample labeled "Aug. 23/78 top ash").
- BDS-5      Sample of upper ash slurry collected at end of ash line on August 23, 1978.

BDS-6      Sample of bottom ash slurry collected at end of ash line on August 17, 1978.

BDS-7      Sample of bottom ash slurry collected at end of ash line on August 18, 1978.

BDS-8      Sample of bottom ash slurry collected at end of ash line on August 21, 1978.

BDS-9      Sample of bottom ash slurry collected at end of ash line probably on August 22, 1978 (labeled "Aug 23/78 bottom ash end of ash line").

BDS-10     Sample of bottom ash slurry collected at end of ash line on August 23, 1978.

NOTE: No slurry samples were taken on August 19 or August 20, and one sample of fly ash slurry and of bottom ash slurry were mislabeled.

BDW-1     Sample of ash lagoon overflow collected on August 17, 1978. (Minor solids present in sample--should be filtered.)

BDW-2     Sample of ash lagoon overflow collected on August 18, 1978. (Minor solids present in sample--should be filtered.)

BDW-3     Sample of ash lagoon overflow collected on August 21, 1978. (Minor solids present in sample--should be filtered.)

BDW-4     Sample of ash lagoon overflow probably collected on August 22, 1978. (Minor solids present in sample--should be filtered.)

BDW-5     Sample of ash lagoon overflow collected on August 23, 1978. (Minor solids present in sample--should be filtered.)

BDW-6     Sample of slurry water collected on August 17, 1978. (Minor solids present in sample--should be filtered.)

BDW-7     Sample of slurry water collected on August 18, 1978. (Minor solids present in sample--should be filtered.)

BDW-8     Sample of slurry water collected on August 21, 1978. (Minor solids present in sample--should be filtered.)

BDW-9     Sample of slurry water collected on August 22, 1978. (Minor solids present in sample--should be filtered.)

BDW-10    Sample of slurry water collected on August 23, 1978. (Minor solids present in sample--should be filtered.)

NOTE: No slurry water samples were collected on August 19 or August 20.

BOUNDARY DAM G.S. UNIT #6 ASH LAGOON STUDY

ASH LEACHATE TEST RESULTS  
Environmental Protection Service  
Environment Canada  
February, 1979

Introduction

An ash lagoon water quality study was conducted at the Boundary Dam G.S. Unit #6 for the Poplar River G.S. to check the assumptions being made on the transferability of leachate test results to ash lagoon water quality. Twice daily on August 17, 18, 21, 22 and 23, 1978 samples were taken by the Saskatchewan Power Corporation (SPC) at Boundary Dam of each of the following:

- (i) intake slurry water (reservoir water)
- (ii) pulverized coal
- (iii) bottom ash (wet)
- (iv) flyash (dry)
- (v) bottom ash slurry
- (vi) flyash slurry
- (vii) lagoon water

A full set of samples was provided to Dr. Pagenkopf of the University of Montana for analysis and ash leaching experiments. Environment Canada also obtained splits of the composited 5-day samples of bottom ash and flyash, and sufficient reservoir water was provided by the SPC to conduct ash leachate tests at the ash to water ratios used in actual operation.

Preparation of Leachate Solutions

The leaching solutions were prepared at an ash to water ratio of 55,000 mg/l (the same ratio used in the University of Montana experiments) and where composite ash was required, it consisted of 42% bottom ash and 52% flyash by weight. The solutions were, however, set up somewhat differently than in the University of Montana experiments. Each solution was set up for a specific leaching period of 7, 14 or 21 days so that only one aliquot would have to be drawn from each solution, that being at the end of one of the specified periods. This procedure was adopted to prevent changes in the ash to water ratio that would occur if aliquots were taken successively from a larger sample. The samples set up in triplicate for the three time periods are identified as follows:

<u>Sample #</u>	<u>Composition</u>	<u>Time Period for Leaching</u>
1	Flyash	7 days
2	Bottom ash	7 days
3	Composite	7 days
4	Flyash	14 days
5	Bottom ash	14 days
6	Composite	14 days
7	Flyash	21 days
8	Bottom ash	21 days
9	Composite	21 days

Each sample consisted of 22 g ash to 400 ml of water. The ash was first mixed with 300 ml of reservoir water and shaken in a polypropylene bottle for ten minutes. The mixture was then transferred to a polypropylene leaching container and the bottle rinsed with the remaining 100 ml of reservoir water. The sediments were allowed to settle and the pH and conductivity measured. The tests were initiated November 27, 1978 and the initial readings were as follows:

<u>Sample #</u>	<u>pH</u>	<u>Conductivity (millimhos)</u>
1	11.2	0.93
2	8.8	0.76
3	10.6	0.73
4	11.2	0.96
5	8.83	0.73
6	10.6	0.75
7	11.0	0.89
8	8.83	0.75
9	10.73	0.78

The samples were allowed to stand uncovered for two days and the measurements repeated on November 29, 1978 were as follows:

<u>Sample #</u>	<u>pH</u>	<u>Conductivity (millimhos)</u>
1	10.75	0.75
2	8.83	0.84
3	10.33	0.79
4	10.90	0.80
5	8.85	0.76
6	10.37	0.75
7	10.77	0.74
8	8.85	0.79
9	10.30	0.74

The samples were then covered and sampled at seven day intervals. The volume, pH and conductivity of each sample withdrawn was measured and the sample was then filtered through 0.45 micron filter paper prior to shipping for analysis. The results are as follows:

<u>Date Sampled</u>	<u>Sample #</u>	<u>Volume (ml)</u>	<u>pH</u>	<u>Conductivity (millimhos)</u>
Dec. 4/78	1	300	10.1	0.75
Dec. 4/78	2	300	8.93	0.76
Dec. 4/78	3	275	9.67	0.89
Dec. 11/78	4	300	10.03	0.70
Dec. 11/78	5	300	8.90	0.80
Dec. 11/78	6	250	9.32	0.96
Dec. 18/78	7	270	9.50	0.86
Dec. 18/78	8	300	8.87	0.84
Dec. 18/78	9	275	9.27	0.83

Samples of the reservoir water submitted December 21, 1978 for analysis had the following values:

<u>Sample #</u>	<u>pH</u>	<u>Conductivity (millimhos)</u>
1	7.8	0.66
2	7.8	0.66
3	7.83	0.68

#### Results of Analysis

Three separate laboratories received equal splits of samples 1 through 9 as well as a reservoir water sample for analysis. The laboratories receiving samples are:

Technitrol Canada Limited  
 Barringer-Magenta Limited  
 Technical Services Laboratories

The results of the analysis conducted by each of the laboratories are appended.

H-6024  
Method

	F.A. 4,12,78	B.A. 4,12,78	Mix FA & BA 4,12,78	B.A. 11,12,78	F.A. 11,12,78	Mix FA & BA 11/12/78	B.A. 11/12/78	F.A. 11/12/78	Mix Atomic Absorption
Calcium	6.9 ppm	8.6 ppm	17.7 ppm	24.3 ppm	25 ppm	63.6 ppm	"	"	
Magnesium	0.15 ppm	14.3 ppm	0.42 ppm	0.22 ppm	25.6 ppm	0.97 ppm	"	"	
Sodium	126.9 ppm	103.8 ppm	125.6 ppm	136.2 ppm	112.8 ppm	150 ppm	"	"	
Potassium	15.3 ppm	9.5 ppm	15.8 ppm	11.2 ppm	10 ppm	12.2 ppm	"	"	
Chloride	34 ppm	30 ppm	32 ppm	25 ppm	16 ppm	18 ppm			
Nitrate NO <sub>3</sub>	2.5 ppm	4.1 ppm	2.1 ppm	7.5 ppm	0.35 ppm	4.43 ppm			
Sulphate SO <sub>4</sub>	186 ppm	184.4 ppm	266.7 ppm	234.6 ppm	195.5 ppm	407.4 ppm			
Alkalinity	140 ppm	170 ppm	160 ppm	140 ppm	180 ppm	80 ppm			
SiO <sub>2</sub>	1 ppm	8 ppm	2 ppm	3 ppm	8 ppm	1 ppm			
Aluminum	15.5 ppm	1.2 ppm	15.5 ppm	11.2 ppm	1.3 ppm	13.4 ppm			
Arsenic	<10 ppb	<10 ppb	<10 ppb	<10 ppb	<10 ppb	<10 ppb			
Boron	11.2 ppm	3.4 ppm	14.4 ppm	11.3 ppm	3.6 ppm	19.5 ppm			
Cadmium	0.01 ppm	0.01 ppm	0.02 ppm	<0.01 ppm	0.3 ppm	<0.01 ppm			
Copper	<0.01 ppm	0.10 ppm	<0.01 ppm	0.095 ppm	0.095 ppm	0.14 ppm			
Chrome	0.4 ppm	0.2 ppm	0.2 ppm	0.16 ppm	0.08 ppm	0.16 ppm			
Fluoride	1.5 ppm	1.8 ppm	2.5 ppm	<1.0 ppm	<1.0 ppm	<1.0 ppm			
Iron	0.04 ppm	0.04 ppm	0.08 ppm	0.08 ppm	0.04 ppm	0.04 ppm			
Manganese	<0.01 ppm	0.18 ppm	<0.01 ppm	0.18 ppm	<0.01 ppm	<0.01 ppm			
Lead	<0.01 ppm	<0.01 ppm	1.8 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm			
Selenium	2.1 ppm	3.0 ppm	3.0 ppm	1.7 ppm	0.02 ppm	<0.01 ppm			
Beryllium	<0.01 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm			
Zinc	<0.01 ppm	1.2 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm			
DOC	30 ppm	27 ppm	33 ppm	17 ppm	19 ppm	19 ppm	Dornan DC=50		

	F.A. 18/12/78	B.A. 18/12/78	Mix 18/12/78	FA & BA 18/12/78	Boundary Dam Reservoir Water (F)	Method
Calcium	76.9 ppm	63.2 ppm	97.9 ppm	44.9 ppm		Atomic Absorption
Magnesium	0.68 ppm	9.4 ppm	1.3 ppm	24.8 ppm	" "	
Sodium	130 ppm	121.3 ppm	108.9 ppm	77.1 ppm	" "	
Potassium	14.3 ppm	12.7 ppm	13.8 ppm	11.8 ppm	" "	
Chloride	24 ppm	14 ppm	17 ppm	242 ppm	Titration	
Nitrate NO <sub>3</sub>	8.8 ppm	6.8 ppm	8.8 ppm	7.03 ppm	Spectrophotometric	
Sulphate SO <sub>4</sub>					Gravimetric	
Alkalinity	160 ppm	200 ppm	180 ppm	194.2 ppm	Titration	
SiO <sub>2</sub>	1 ppm	10 ppm	1 ppm	190 ppm	Atomic Absorption	
Aluminum	8.3 ppm	1.4 ppm	8.3 ppm	3 ppm	Atomic Absorption	
Arsenic	<10 ppb	<10 ppb	<10 ppb	<0.01 ppm	Spectrophotometric	
Boron	17.1 ppm	3.8 ppm	16.6 ppm	<10 ppb	Curcumin Method	
Cadmium	<0.01 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm	Atomic Absorption	
Copper	0.05 ppm	0.1 ppm	<0.01 ppm	<0.01 ppm	" "	
Chrome	<0.01 ppm	<0.01 ppm	0.1 ppm	<0.01 ppm	" "	
Fluoride	<1.0 ppm	<1.0 ppm	<1.0 ppm	1.0 ppm	Spectrophotometric	
Iron	0.19 ppm	0.05 ppm	0.24 ppm	0.04 ppm	Atomic Absorption	
Manganese	<0.01 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm	" "	
Lead	<0.01 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm	" "	
Selenium	<0.01 ppm	<0.01 ppm	<0.01 ppm	0.41 ppm	" "	
Beryllium	<0.01 ppm	<0.01 ppm	<0.01 ppm	<0.01 ppm	" "	
Zinc	0.04 ppm	0.04 ppm	0.09 ppm	0.025 ppm	" "	
DOC	6 ppm	20 ppm	15 ppm	9 ppm	Dornan DC=50	

**TECHNICAL SERVICE LABORATORIES**

DIVISION OF SURGEON TECHNICAL ENTERPRISES LIMITED

1301 FEWSTER DRIVE, MISSISSAUGA, ONT. L4W 1A2

CERTIFICATE OF ANALYSIS

TELEPHONE: 625-1544 - AREA 416

Samples Received December 5, 1978

	F.A. 4/12/78	B.A. 4/12/78	Mix F.A. & B.A. 4/12/78
Calcium (Ca) mg/litre	35.	34.	70.
Magnesium (Mg) mg/litre	0.16	22.	0.72
Sodium (Na) mg/litre	150.	128.	155.
Potassium (K) mg/litre	16.	13.	16.
Chloride (Cl) mg/litre	41.	9.6	.31.
Nitrate (NO <sub>3</sub> ) mg/litre	2.6	1.8	1.7
Sulphate (SO <sub>4</sub> ) mg/litre	277.	295.	386.
Alkalinity, mg/litre	120.	190.	110.
Silicon (Si) mg/litre	0.99	2.8	0.27
Boron (B) mg/litre	10.	2.9	12.
Cation-Anion Balance	1044.	1082.	1285.

Samples Received December 11, 1978

	F.A. 11/12/78	B.A. 11/12/78	Mix F.A. & B.A. 11/12/78
Calcium (Ca) mg/litre	32.	29.	81.
Magnesium (Mg) mg/litre	0.28	27.	1.2
Sodium (Na) mg/litre	148.	148.	171.
Potassium (K) mg/litre	16.	14.	18.
Chloride (Cl) mg/litre	30.	3.2	13.
Nitrate (NO <sub>3</sub> ) mg/litre	3.5	1.0	5.1
Sulphate (SO <sub>4</sub> ) mg/litre	224.	287.	378.
Alkalinity, mg/litre	120.	200.	110.
Silicon (Si) mg/litre	1.6	3.4	0.46
Boron (B) mg/litre	10.	3.5	15.
Cation-Anion Balance	928.	110.	1307.

**TECHNICAL SERVICE LABORATORIES**

DIVISION OF BURGENER TECHNICAL ENTERPRISES LIMITED

1301 FEWSTER DRIVE, MISSISSAUGA, ONT. L4W 1A2

CERTIFICATE OF ANALYSIS

TELEPHONE: 625-1544 - AREA 416

Samples Received December 20, 1978

	F.A. 20/12/78	B.A. 20/12/78	Mix F.A. & B.A. 20/12/78
Calcium (Ca) mg/litre	55.	27.	52.
Magnesium (Mg) mg/litre	0.80	29.	1.4
Sodium (Na) mg/litre	161.	148.	153.
Potassium (K) mg/litre	16.	14.	15.
Chloride (Cl) mg/litre	34.	27.	10.
Nitrate (NO <sub>3</sub> ) mg/litre	2.2	1.3	0.89
Sulphate (SO <sub>4</sub> ) mg/litre	321.	290.	346.
Alkalinity, mg/litre	110.	200.	480.
Silicon (Si) mg/litre	0.74	3.2	0.40
Boron (B) mg/litre	14.	3.6	14.
Cation-Anion Balance	1167.	1173.	1394.

Samples Received January 4, 1979

	#3	#4
Calcium (Ca) mg/litre	40.	39.
Magnesium (Mg) mg/litre	22.	22.
Sodium (Na) mg/litre	96.	96.
Potassium (K) mg/litre	14.	13.
Chloride (Cl) mg/litre	40.	46.
Nitrate (NO <sub>3</sub> ) mg/litre	2.6	2.7
Sulphate (SO <sub>4</sub> ) mg/litre	220.	225.
Alkalinity, mg/litre	200.	190.
Silicon (Si) mg/litre	1.1	1.2
Boron (B) mg/litre	0.31	0.27
Cation-Anion Balance	990.	998.

**TECHNICAL SERVICE LABORATORIES**

DIVISION OF BURGNER TECHNICAL ENTERPRISES LIMITED

1301 FEWSTER DRIVE, MISSISSAUGA, ONT. L4W 1A2

CERTIFICATE OF ANALYSIS

TELEPHONE: 625-1544 - AREA 416

	F.A. 4/12/78	B.A. 4/12/78	MIX FA & BA 4/12/78	F.A. 11/12/78	B.A. 11/12/78	MIX FA & BA 11/12/78
Aluminum (Al) mg/l	17.	0.84	9.4	13.	0.76	14.
Chromium (Cr) mg/l	0.18	<0.05	0.09	0.20	<0.05	0.11
Boron (B) mg/l	10.	2.90	12.	10.	3.5	15.
Cadmium (Cd) mg/l	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Copper (Cu) mg/l	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Iron (Fe) mg/l	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Manganese (Mn) mg/l	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Lead (Pb) mg/l	<0.05	<0.05	<0.05	0.10	0.08	0.07
Selenium (Se) mg/l	0.17	0.22	0.17	0.28	0.22	0.30
Zinc (Zn) mg/l	0.81	0.63	1.4	1.4	0.76	1.90
Floride (F) mg/l	0.62	0.08	Insufficient Sample	0.45	0.11	1.35

	F.A.	B.A.	F.A. & B.A.	#3	#3F
Aluminum (Al) mg/l	17.	0.84	8.5	0.50	0.50
Chromium (Cr) mg/l	0.23	<0.05	0.07	<0.05	<0.05
Boron (B) mg/l	14.	3.6	14.	0.31	0.27
Cadmium (Cd) mg/l	<0.01	<0.01	<0.01	<0.01	<0.01
Copper (Cu) mg/l	<0.05	<0.05	<0.05	<0.05	<0.05
Iron (Fe) mg/l	<0.05	<0.05	<0.05	<0.05	<0.05
Manganese (Mn) mg/l	<0.05	<0.05	<0.05	<0.05	<0.05
Lead (Pb) mg/l	<0.05	<0.05	<0.05	<0.05	<0.05
Selenium (Se) mg/l	0.26	0.24	0.38	0.30	0.32
Zinc (Zn) mg/l	1.1	<0.05	1.1	0.39	0.76
Floride (F) mg/l	0.61	0.12	0.95	0.15	0.15

REPORT ON THE ANALYSIS OF ELEVEN (11)  
WATER SAMPLES AND ASH LEACHATES  
FISHERIES AND ENVIRONMENT CANADA  
CONTRACT NO. KE204-8-1507

PREPARED FOR  
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FEBRUARY, 1979

This report covers the analyses performed on a total of two reservoir waters and three each of bottom ash, fly ash and combined ash leachate waters, as requested by the Fisheries and Environment Canada, Contract No. KE204-8-1507.

The analytical results are enclosed under our work order numbers 78-986, 78-1002, 78-1019 and 79-1. Please note that the analyses for specific conductivity, Doc and Phenols were not performed.

All metals, with the exceptions of As, Se, Cd, Cu, Pb and Zn, were analysed by our multi-element plasma emission system on an acidified sample which underwent a tenfold preconcentration. Cd, Cu, Pb and Zn were determined, on the same solutions, by atomic absorption spectroscopy. The determinations of As and Se were done using a hydride generator with atomic absorption employing heated quartz tube atomization. The anions were determined using a Dionex Model 10 Ion Chromatograph under standard conditions. The alkalinity was done titrimetrically using .02N  $H_2SO_4$  as the titrant. All results are quoted as  $\mu g/l$  in the original sample. Silicon was quoted as  $\mu g/l$  Si. To convert these results to  $\mu g/l$   $SiO_2$ , a conversion factor of 2.139 must be applied.

A cation-anion balance was done on the results obtained, and the resultant values are given in Fig. 1. The metals which contributed significantly to the total meq/l determination were Al(III), Ca(II), Cr(VI), Fe(III), Mg(II), Mo(VI), K(I), Ba(II), Si(IV), Na(I), Sr(II) and V(V). The anions considered were  $F^-$ ,  $Cl^-$ ,  $PO_4^{=}$ ,  $NO_3^-$ ,  $SO_4^{=}$ ,  $OH^-$ ,  $CO_3^{=}$  and  $HCO_3^-$ . A second figure is quoted as Anion Equivalent II. This value incorporates the boron as  $BO_3^{=}$ . It should be noted, however, that in aqueous solution, the boron is probably in the form of the very weakly

dissociated boric acid. Being an extremely weak acid ( $K = 5.8 \times 10^{-10}$ ) it would not contribute significantly to the balance, as illustrated by the off-balance towards the anion side when  $\text{BO}_3^{\frac{3}{2}-}$  is included in the calculation.

FIG. 1  
CATION-ANION BALANCE CALCULATION

<u>Sample</u>	<u>Cation Equivalents (meq/l)</u>	<u>Anion Equivalents I (meq/l)</u>	<u>Anion Equivalents II (meq/l)</u>
F.A. #1	9.89	7.66	10.43
B.A. #2	8.57	8.69	9.43
Mix F.A.&B.A. #3	10.86	9.85	12.62
F.A. #4	8.29	7.22	10.14
B.A. #5	8.65	9.08	9.99
F.A. & B.A. #6	11.17	10.59	14.48
F.A. #7	8.88	9.21	12.70
B.A. #8	7.96	9.18	10.16
Mix F.A.&B.A. #9	9.87	9.70	13.40
Boundary Dam			
Reservoir Water #1	7.08	7.78	7.88
Boundary Dam			
Reservoir Water			
#Fl	6.85	7.79	7.89

# BARRINGER RESEARCH LIMITED

304 CARLINGVIEW DRIVE  
REXDALE, ONTARIO, CANADA  
PHONE: 416-677-2491  
CABLE: BARESEARCH

Dr. D.W. Draper

## Geochemical Laboratory Report

DATE February 23, 1979

Environmental Protection Services  
Power Generation & Energy Div.  
Abatement & Compliance Branch  
Water Pollution Control Directorate  
351 St. Joseph Blvd., 13th fl., Place Vincent Massey  
Hull, Quebec, K1A 1C8  
REPORT NUMBER 78-986

Contract #KE204-8-1507

SAMPLE NUMBER	Al µg/l	As µg/l	Be µg/l	Ca µg/l	Cd µg/l	Cr µg/l	Co µg/l	Cu µg/l	Fe µg/l	Pb µg/l
F.A. #1	13200	.6	< .5	38000	< .2	158	7	4	25	< 2
B.A. #2	478	11.2	< .5	37200	< .2	16	< 5	4	5	< 2
Mix F.A. & B.A.#3	12100	2.8	< .5	68700	< .2	80	< 5	3	26	< 2
	Mg µg/l	Mn µg/l	Mo µg/l	Ni µg/l	K µg/l	Ba µg/l	Se µg/l	Si µg/l	Ag µg/l	Na µg/l
F.A. #1	< 5	< 1	290	< 5	13300	82.3	47.5	1690	< .5	132000
B.A. #2	14400	< 1	< 20	6	10500	51.4	2.6	1190	< .5	115000
Mix F.A.&B.A. #3	448	< 1	270	< 5	390	< .2	47.5	647	< .5	133000
	Sr µg/l	B µg/l	Sn µg/l	Ti µg/l	Zr µg/l	V µg/l	Zn µg/l			
F.A. #1	6900	9980	< 10	< .5	< .5	38.1	5			
B.A. #2	1210	2690	< 10	< .5	< .5	42.1	9			
Mix F.A.&B.A. #3	5690	9990	< 10	< .5	.5	56.4	2			
	F <sup>-</sup> µg/l	Cl <sup>-</sup> µg/l	PO <sub>4</sub> <sup>3-</sup> µg/l	NO <sub>3</sub> <sup>-</sup> µg/l	SO <sub>4</sub> <sup>=</sup> µg/l	OH <sup>-</sup> Alkalin. µg/l *	CO <sub>3</sub> <sup>=</sup> Alkal. µg/l *	HCO <sub>3</sub> <sup>-</sup> Alkal. µg/l *	pH	
F.A. #1	2690	12000	< 100	660	233000	0	84000	32000	8.9	
B.A. #2	237	10800	< 100	< 100	233000	0	0	176000	7.7	
Mix F.A.&B.A. #3	3210	12200	< 100	990	336000	0	44000	72000	8.6	

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## Laboratory Report

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CABLE: BARESEARCH

DATE February 23, 1979

REPORT NUMBER 78-1002

Contract KE204-8-1507

SAMPLE NUMBER	Al μg/l	As μg/l	Be μg/l	Ca μg/l	Cd μg/l	Cr μg/l	Co μg/l	Cu μg/l	Fe μg/l	Pb μg/l
F.A. #4	9510	3.8	<.5	31600	<.2	163	<5	8	12	<2
B.A. #5	337	14.4	<.5	30000	<.2	14	<5	.3	3	<2
F.A. & B.A. #6	9080	5.6	<.5	75600	<.2	83	<5	4	61	<2
	Mg μg/l	Mn μg/l	Mo μg/l	Ni μg/l	K μg/l	Ba μg/l	Se μg/l	Si μg/l	Ag μg/l	Na μg/l
F.A. #4	201	<1	180	<5	12300	81.5	45.0	1450	<.5	114000
B.A. #5	16600	1	<20	<5	11300	58.4	3.4	3290	<.5	114000
F.A. & B.A. #6	887	.1	170	<5	14000	125	43.0	331	<.5	132000
	Sr μg/l	B μg/l	Sn μg/l	Ti μg/l	Zr μg/l	V μg/l	Zn μg/l			
F.A. #4	5370	10500	<10	<.5	<.5	43.6	10			
B.A. #5	1120	3290	<10	<.5	<.5	40.9	3			
F.A. & B.A. #6	5500	14000	<10	<.5	<.5	48.4	500			
	F <sup>-</sup> μg/l	Cl <sup>-</sup> μg/l	PO <sub>4</sub> <sup>=</sup> μg/l	NO <sub>3</sub> <sup>-</sup> μg/l	SO <sub>4</sub> <sup>=</sup> μg/l	OH <sup>-</sup> Alkalin. μg/l *	CO <sub>3</sub> <sup>=</sup> Alkal. μg/l *	HCO <sub>3</sub> <sup>-</sup> Alkal. μg/l *	pH	
F.A. #4	2020	11600	<100	1740	211000	0	112000	6000	9.2	
B.A. #5	173	12300	<100	<100	252000	0	0174000	8.2		
F.A. & B.A. #6	2680	14700	<100	1070	362000	0	68000	56000	8.7	

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## Geochemical Laboratory Report

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DATE February 23, 1979

REPORT NUMBER 78-1019 Contract KE204-8-1507

SAMPLE NUMBER	Al μg/l	As μg/l	Be μg/l	Ca μg/l	Cd μg/l	Cr μg/l	Co μg/l	Cu μg/l	Fe μg/l	Pb μg/l
F.A. #7	4020	5.6	< .5	56600	< .2	204	< 5	5	26	< 2
B.A. #8	296	14.4	< .5	29200	< .2	18	< 5	3	16	< 2
Mix F.A.&B.A. #9	4210	6.8	< .5	77200	< .2	66	< 5	4	57	< 2
	Mg μg/l	Mn μg/l	Mo μg/l	Ni μg/l	K μg/l	Ba μg/l	Se μg/l	Si μg/l	Ag μg/l	Na μg/l
F.A. #7	470	< 1	260	< 5	11400	108	23.0	170	< .5	117000
B.A. #8	18500	1	< 20	< 5	9620	25.1	3.2	1660	< .5	102000
Mix F.A.&B.A. #9	1420	27	210	< 5	11100	58.3	26.0	309	< .5	114000
	Sr μg/l	B μg/l	Sn μg/l	Ti μg/l	Zr μg/l	V μg/l	Zn μg/l			
F.A. #7	5130	12600	10	< .5	< .5	33.1	7			
B.A. #8	1050	3520	20	< .5	< .5	36.4	5			
Mix F.A.&B.A. #9	5170	13300	<10	< .5	< .5	38.4	6			
	F <sup>-</sup> μg/l	Cl <sup>-</sup> μg/l	PO <sub>4</sub> <sup>=</sup> μg/l	NO <sub>3</sub> <sup>-</sup> μg/l	SO <sub>4</sub> <sup>=</sup> μg/l	OH <sup>-</sup> Alkal. μg/l*	CO <sub>3</sub> <sup>=</sup> Alkal. μg/l*	HCO <sub>3</sub> <sup>-</sup> Alkal. μg/l *	pH	
F.A. #7	1830	12100	< 100	1490	295000	0	120000	10000	9.2	
B.A. #8	207	12200	< 100	200	246000	0	20000	165000	8.4	
Mix F.A.&B.A. #9	1950	12100	< 100	< 100	320000	0	80000	50000	8.9	

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## Geochemical Laboratory Report

304 CARLINGVIEW DRIVE  
 REXDALE, ONTARIO, CANADA  
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 CABLE: BARESEARCH

DATE February 23, 1979

REPORT NUMBER 79-1

Contract KE204-8-1507

SAMPLE NUMBER	Al µg/l	As µg/l	Be µg/l	Ca µg/l	Cd µg/l	Cr µg/l	Co µg/l	Cu µg/l	Fe µg/l	Pb µg/l
Boundary Dam Reservoir Water #1	58	10.0	< .5	44600	< .2	15	< 5	9	17	< 2
#Fl	45	10.0	< .5	43200	< .2	15	< 5	12	16	< 2
	Mg µg/l	Mn µg/l	Mo µg/l	Ni µg/l	K µg/l	Ba µg/l	Se µg/l	Si µg/l	Ag µg/l	Na µg/l
Boundary Dam Reservoir Water #1	16000	4	< 20	6	9490	125	.8	862	< .5	72400
#Fl	15900	< 1	< 20	6	9270	67	.4	420	< .5	70500
	Sr µg/l	B µg/l	Sn µg/l	Ti µg/l	Zr µg/l	V µg/l	Zn µg/l			
Boundary Dam Reservoir Water #1	357	355	<10	2.5	11.3	4.6	3			
#Fl	334	362	<10	<.5	< .5	4.4	7			
	F <sup>-</sup> µg/l	Cl <sup>-</sup> µg/l	PO <sub>4</sub> <sup>=</sup> µg/l	NO <sub>3</sub> <sup>-</sup> µg/l	SO <sub>4</sub> <sup>=</sup> µg/l	OH <sup>-</sup> Alkal. µg/l *	CO <sub>3</sub> <sup>=</sup> Alkal. µg/l *	HCO <sub>3</sub> <sup>-</sup> Alkal. µg/l *	pH	
Boundary Dam Reservoir Water #1	270	9520	180	940	190000	0	0	176000	7.9	
#Fl	257	9760	140	990	190000	0	0	176000	7.9	

ANNEX G

ASH LAGOON SEEPAGE STUDY



HYDROLOGIC IMPACT OF ASH LAGOONS  
AT CORONACH, SASKATCHEWAN

Prepared for:

The International Poplar River  
Water Quality Board

March 1979

By:

Geraghty & Miller, Inc.  
Consulting Ground-Water Geologists and Hydrologists  
Annapolis, Maryland 21401

## CONTENTS

	<u>Page</u>
INTRODUCTION	G-5
CONCLUSIONS	G-8
THE HYDROLOGIC SYSTEM	G-10
Climate	G-10
The Surface-Water System	G-10
The Ground-Water System	G-11
The Ground-Water Flow Regime	G-17
ASH-DISPOSAL LAGOON OPERATION	G-26
IMPACTS OF ASH-DISPOSAL LAGOONS	G-29
Seepage to the Ground-Water System	G-29
Seepage Pathways and Destinations	G-35
CONTAMINATION ABATEMENT ALTERNATIVES	G-54
REFERENCES	G-56

## FIGURES

<u>Figure</u>		<u>Page</u>
1	Site Location	G-6
2	West to East Stratigraphic Cross Section of the Proposed Ash-Lagoon Location	G-16
3	Potentiometric Level of the Frenchman Aquifer, Fall 1978	G-18
4	Potentiometric Levels of the Hart Coal Aquifer	G-20
5	Water-Level Fluctuations in the Hart Coal Aquifer and Cookson Reservoir, November 1975 to June 1978	G-21
6	Water-Level Fluctuation in the Empress Group, March 12 to April 12, 1978	G-23
7	Potentiometric Level of the Empress Group, June 1978	G-24
8	Potentiometric Level of the Empress Group, January 1979	G-25
9	Proposed Ash-Lagoon Location	G-31
10	Vertical Flow of Ground Water in the Till and Empress Group, June 1978	G-36
11	North to South One-to-One Scale Cross Section	G-38
12	Water-Level Changes in the Empress Group Resulting from 0.01 m <sup>3</sup> /sec Leachate Discharge	G-41
13	Water-Level Changes in the Empress Group Resulting from 0.1 m <sup>3</sup> /sec Leachate Discharge	G-42
14	Scenario 1 - Leachate Flow Pattern	G-43
15	Scenario 2 - Leachate Flow Pattern	G-44
16	Scenario 3 - Leachate Flow Pattern	G-45
17	Scenario 4 - Leachate Flow Pattern	G-46
18	Scenario 5 - Leachate Flow Pattern	G-47
19	Scenario 6 - Leachate Flow Pattern	G-48

TABLES

<u>Table</u>	<u>Page</u>
1 Aquifer Hydraulic Properties	G-12
2 Operating Conditions - Ash Lagoons	G-27
3 Ash Lagoon Scenarios	G-30
4 Leachate Seepage Rates	G-34
5 Length of the East Poplar River Affected by Leachate	G-50

## INTRODUCTION

In mid-January 1979, Geraghty & Miller, Inc., (hereinafter referred to as the Consultant) was retained by the International Poplar River Water Quality Board (IPRWQB) to assess the impact of seepage from ash-disposal lagoons on the hydrologic system. The proposed lagoons are at the Saskatchewan Power Corporation's (SPC) Coronach electricity-generating facility located approximately 5 km (kilometres) north of the International Border between the United States and Canada (Figure 1). The principal objectives of the study were to estimate the volumes of leachate entering the ground-water system from the ash-disposal lagoons, and to determine the ultimate fate of the leachate, i.e., the estimated proportions reaching the surface-water environment in Cookson Reservoir, the East Poplar River, or other local surface-water bodies. Six alternative scenarios concerning potential lagoon location or operation were presented for evaluation. In addition, the Consultant was requested to suggest alternative actions to prevent or minimize the impacts of the lagoons on the ground- and surface-water systems.

This investigation, undertaken during a period between January 10, 1979, and January 31, 1979, is based on available data in the files of Environment Canada. The data include reports by Saskmont Engineering Company Limited

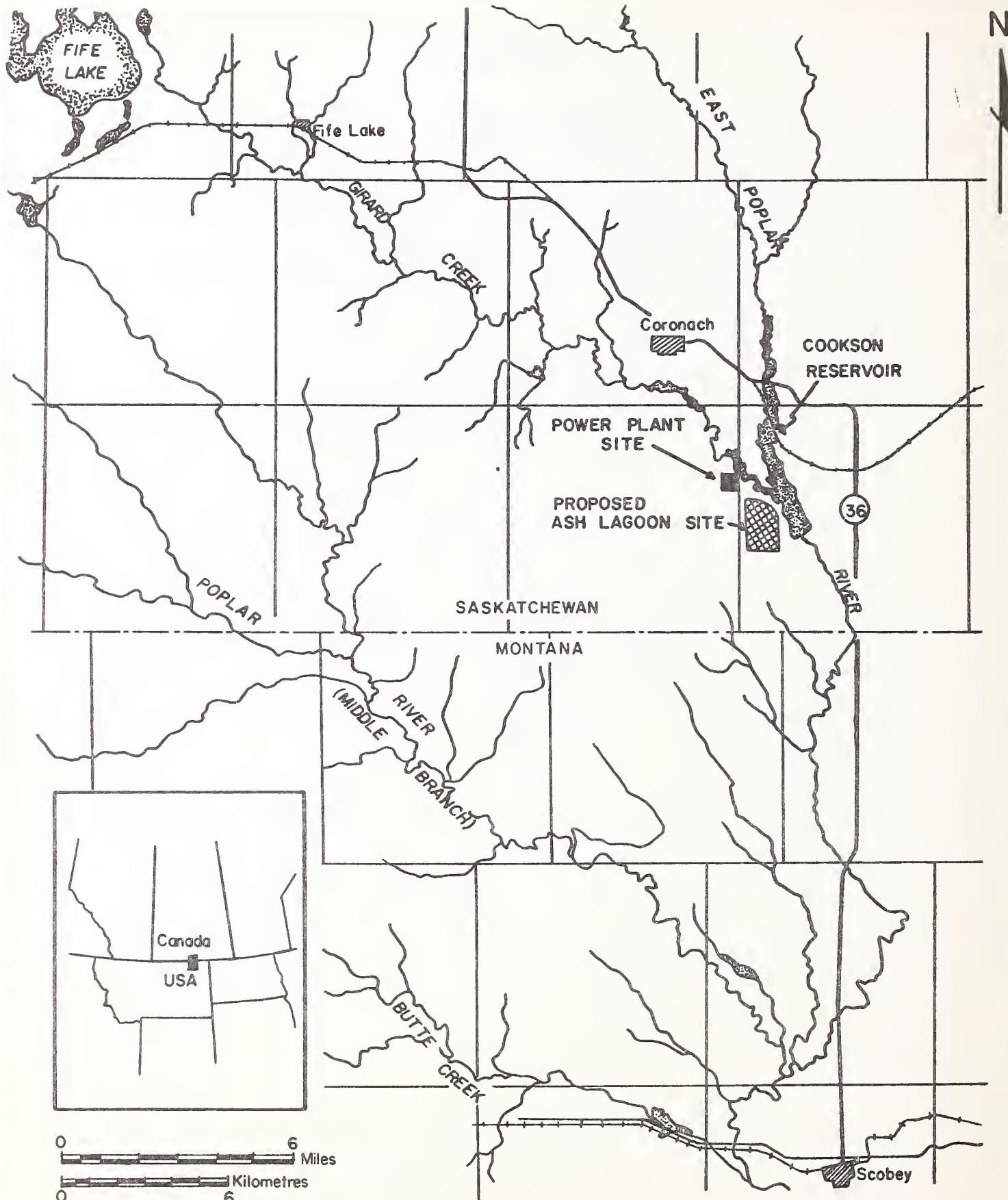


Figure 1. Site Location.

and Ground Engineering Limited done for SPC, Silverspoon Research and Consulting Limited done for the Committee on Ground-Water Quantity and Quality (GWQQC) of the IPRWQB, and the GWQQC. No site visits were made and no original data was collected by the Consultant. Wayne Nordquist of SPC provided the Consultant with water-level readings collected on January 23, 1978, from piezometers constructed into the Empress Group and the glacial till in the immediate vicinity of the currently proposed lagoon site.

Presented herein is the Consultant's best judgment as to the fate of the ash-disposal lagoon leachate based on the available data and his understanding and experience in similar investigations under comparable hydrogeologic conditions. There was insufficient time to allow for the use of computer modeling or other sophisticated analytical methods to refine the conclusions of the report. Further, the Consultant believes that additional field data will be needed if the IPRWQB requires a more definitive solution to the problem of seepage from the ash-disposal lagoon.

The draft report was submitted to the IPRWQB for review on January 31, 1979. This final report takes into account the comments of the IPRWQB, the Project Officer, and SPC. This investigation was made under the supervision of William E. Thompson, Senior Hydrogeologist, with the assistance of James J. Geraghty, President; Vincent W. Uhl, Jr.; and Peter L. Palmer, all of Geraghty & Miller, Inc.

## CONCLUSIONS

1. The rates of seepage from the ash-disposal lagoons is controlled by the (1) head difference between the lagoons and the Empress Group, (2) the hydraulic conductivity of the till and ash, and (3) the area of the lagoons. Reported values for some of these factors, especially hydraulic conductivity of the till, are highly variable, subsequently affecting calculated seepage rates.
2. Seepage from the ash-disposal lagoons will move essentially vertically downward to the Empress Group through the till. Minimal amounts of seepage may move through fractures to the cooling-water drainage canal and small surface ponds and marshes.
3. Seepage into the Empress Group will modify the flow pattern of the Empress Group slightly, but the modification should not be sufficient to change the regional flow patterns.
4. Leachate entering the Empress Group will move south/southeast and discharge into the East Poplar River between Morrison Dam and the International Border.
5. Hydraulic heads in the aquifers beneath the Empress Group are such that significant downward leakage of leachate to these aquifers is not expected.

6. If double liners or dewatering of the ash slurry are impractical as methods to reduce seepage to the ground-water system, a system of pump-back wells could be installed in the Empress Group to intercept the flow of contaminated ground water.

## THE HYDROLOGIC SYSTEM

### Climate

The climate of Coronach, Saskatchewan, can be described as a dry, steppe type with rather severe annual temperature fluctuations characterized by cold winters. Although somewhat evenly distributed, precipitation is greatest during spring and fall and averages about 38 cm/yr (centimetres per year) (Saskmont Engineering, 1978). Of this total, approximately 10 percent (4 cm) infiltrates to recharge the ground-water system. Although evapotranspiration rates are unknown, potential evaporation from open bodies of water such as the lagoons is approximately 106 cm/yr (Saskmont Engineering, 1978).

### The Surface-Water System

The study area is drained southeast to the Missouri River in Montana by the East Poplar River and its tributaries. Flow in the river system is highly variable due to climatic controls. Periods of peak flow generally occur in late spring as a result of snow-melt runoff and rainfall. Periods of low flow are usually in the fall and winter.

The Cookson Reservoir, which provides a source of cooling water for the SPC facility located on the west banks of the reservoir, was constructed on the East Poplar River

below its confluence with Girard Creek. In addition to surface-water flow changes, the reservoir has an impact on the ground-water system. The primary ground-water impact is a result of a new recharge source for the shallow ground-water system. Ground-water flow directions around the reservoir have, therefore, been substantially modified. These modifications and their effects on seepage from the lagoons and leachate destinations will be discussed in greater detail in the following sections.

#### The Ground-Water System

For the purposes of this investigation, only the upper ground-water system is of concern. This system includes, from oldest to youngest, the Frenchman Aquifer, the Lower Ravenscrag Aquifer, the Hart Coal Aquifer, the Upper Ravenscrag Aquifer, the Empress Group, the glacial till, and the valley-fill alluvium. In general, the chemical quality of water in the aquifers is good; however, locally the concentration of some constituents may be abnormally high. Table 1 summarizes the hydraulic properties of the aquifers in the study area.

The Frenchman Aquifer is composed primarily of sand and silt of deltaic origin. Although lenticular, the aquifer has lateral continuity over the entire study area and averages 55 m (metres) in thickness. Hydraulic properties of the aquifer have been reported to be quite variable.

Table 1. Aquifer Hydraulic Properties

Aquifer	Frenchman	Lower Ravenscrag	Hart Coal	Upper Ravenscrag	Empress	Till	Alluvium
<u>Regional 1/</u>							
Thickness (m)	55	50	3.3	38	---	0-25	0-60
Horizontal Hydraulic Conductivity (cm/sec)	$1.4 \times 10^{-3}$ to $5.8 \times 10^{-3}$	$1.1 \times 10^{-5}$ to $8.6 \times 10^{-4}$	$1.6 \times 10^{-2}$ to $2.3 \times 10^{-2}$	$1.1 \times 10^{-4}$ to $1.1 \times 10^{-3}$	$1.2 \times 10^{-1}$	$1.2 \times 10^{-8}$ to $1.1 \times 10^{-4}$	$5.1 \times 10^{-2}$ to $1.1 \times 10^{-1}$
Vertical Hydraulic Conductivity (cm/sec)	$3.5 \times 10^{-7}$ to $1.1 \times 10^{-4}$	$2.2 \times 10^{-7}$ to $1.2 \times 10^{-6}$	$4.6 \times 10^{-3}$	$1.1 \times 10^{-6}$	$1.2 \times 10^{-2}$	$8.5 \times 10^{-5}$	$5.1 \times 10^{-3}$ to $1.1 \times 10^{-2}$
Transmissivity (m <sup>2</sup> /day)	74 to 275	0.5 to 37	33 to 50	3 to 37	200	0.004	500-3000
Storage Coefficient	$2.8 \times 10^{-4}$	$1.0 \times 10^{-3}$	$4.0 \times 10^{-3}$	$1.0 \times 10^{-3}$	---	$5.0 \times 10^{-2}$	$2.0 \times 10^{-1}$
<u>Site Specific 2/</u>							
Thickness (m)	---	---	---	2	15	---	---
Average Hydraulic Conductivity (cm/sec)	---	---	---	$4.0 \times 10^{-5}$	$5.2 \times 10^{-5}$	---	---
				$6.0 \times 10^{-3}$	$9.0 \times 10^{-5}$		

1/ GWQOC, 1978 and Whitaker and Vonhof, 1978

2/ Ground Engineering, 1978

Horizontal hydraulic conductivity ranges from  $1.4 \times 10^{-3}$  to  $5.8 \times 10^{-3}$  cm/sec (centimetres per second); vertical hydraulic conductivity from  $3.5 \times 10^{-7}$  to  $1.1 \times 10^{-4}$  cm/sec; and transmissivity between 74 and  $275 \text{ m}^2/\text{day}$  (metres square per day) (GWQQC, 1978, and Whitaker and Vonhof, 1978).

The Lower Ravenscrag Aquifer is composed of silt and clay with interspersed sand lenses of marsh and alluvial origin. The sand units are lenticular but may be areally extensive in paleo-alluvial valleys. The unit averages 50 m in thickness. Hydraulic properties of the aquifer are variable: horizontal hydraulic conductivity,  $1.1 \times 10^{-5}$  to  $8.6 \times 10^{-4}$  cm/sec; vertical hydraulic conductivity,  $2.2 \times 10^{-7}$  to  $1.2 \times 10^{-6}$  cm/sec; and transmissivity, 0.5 to  $37 \text{ m}^2/\text{day}$  (GWQQC, 1978, and Whitaker and Vonhof, 1978).

The Hart Coal Aquifer is geologically a part of the Ravenscrag Formation but acts as a separate aquifer in the study area. The unit averages about 3.3 m in thickness and is areally extensive. Hydraulic properties of the aquifer are: horizontal hydraulic conductivity,  $1.6 \times 10^{-2}$  to  $2.3 \times 10^{-2}$  cm/sec; vertical hydraulic conductivity,  $4.6 \times 10^{-3}$  cm/sec; and transmissivity, 33 to  $50 \text{ m}^2/\text{day}$  (GWQQC, 1978, and Whitaker and Vonhof, 1978).

The Upper Ravenscrag Aquifer overlays the Hart Coal Aquifer. This unit is, in general, coarser grained than the Lower Ravenscrag; silt and sand predominate. Hydraulic properties of the Upper Ravenscrag Aquifer are: horizontal hydraulic conductivity,  $1.1 \times 10^{-4}$  to  $1.1 \times 10^{-3}$  cm/sec; vertical hydraulic conductivity,  $1.1 \times 10^{-6}$  cm/sec; and transmissivity, 3 to 37  $\text{m}^2/\text{day}$  (GWQOC, 1978, and Whitaker and Vonhof, 1978).

The Empress Group consists of fluvial sands and gravels and talus from upland slopes. The Empress Group is approximately 2 m thick but is not areally extensive. Located primarily along the East Poplar River and Girard Creek, the Empress Group is in direct hydraulic connection with the Hart Coal Aquifer and valley-fill alluvium in the river channel. Regional hydraulic properties have been reported to be: horizontal hydraulic conductivity,  $1.2 \times 10^{-1}$  cm/sec; vertical hydraulic conductivity,  $1.2 \times 10^{-2}$  cm/sec; and transmissivity,  $200 \text{ m}^2/\text{day}$  (Whitaker and Vonhof, 1978). Tests made in the immediate area of the proposed ash-disposal lagoons show lower hydraulic conductivity measurements than the generalized values referred to above. Hydraulic conductivities from tests of 14 piezometers range from  $4.0 \times 10^{-5}$  to  $6.0 \times 10^{-3}$  cm/sec, with a mean value of  $2.2 \times 10^{-3}$  cm/sec (Ground Engineering, 1978).

The glacial till, which rests on top of the Empress Group, averages 15 m in thickness over most of the study area. Because the lagoons will be resting directly on this till layer, knowledge of its hydraulic properties is very important. The till is reported to be sandy and silty with evidence of vertical fracturing. These fractures can serve as conduits for the passage of ground water and lagoon leachate to the deeper ground-water system. Vertical hydraulic conductivities in similarly fractured till in Saskatchewan have been reported to be as high as  $8.5 \times 10^{-5}$  cm/sec compared with a horizontal hydraulic conductivity range of  $1.2 \times 10^{-8}$  to  $1.1 \times 10^{-4}$  cm/sec. The data collected by Ground Engineering indicates that the hydraulic conductivity in the area of the lagoons is higher than the reported values given above.

Alluvium, which may be up to 60 m thick along the East Poplar River and beneath the reservoir, is in direct hydraulic connection with the Empress Group, the Upper Ravenscrag Aquifer, and the Hart Coal Aquifer. As a result, the reservoir has a major direct impact on the flow regime of these aquifers. An east-to-west cross section through the proposed lagoon site shows the relationship of these aquifer systems (Figure 2). Hydraulic properties of the alluvium are: horizontal hydraulic conductivity,  $5.1 \times 10^{-2}$  to  $1.1 \times 10^{-1}$  cm/sec; vertical hydraulic conductivity,  $5.1 \times 10^{-3}$  to  $1.1 \times 10^{-2}$  cm/sec; and transmissivity, 500 to 3,000  $\text{m}^2/\text{day}$  (GWQQC, 1978, and Whitaker and Vonhof, 1978).

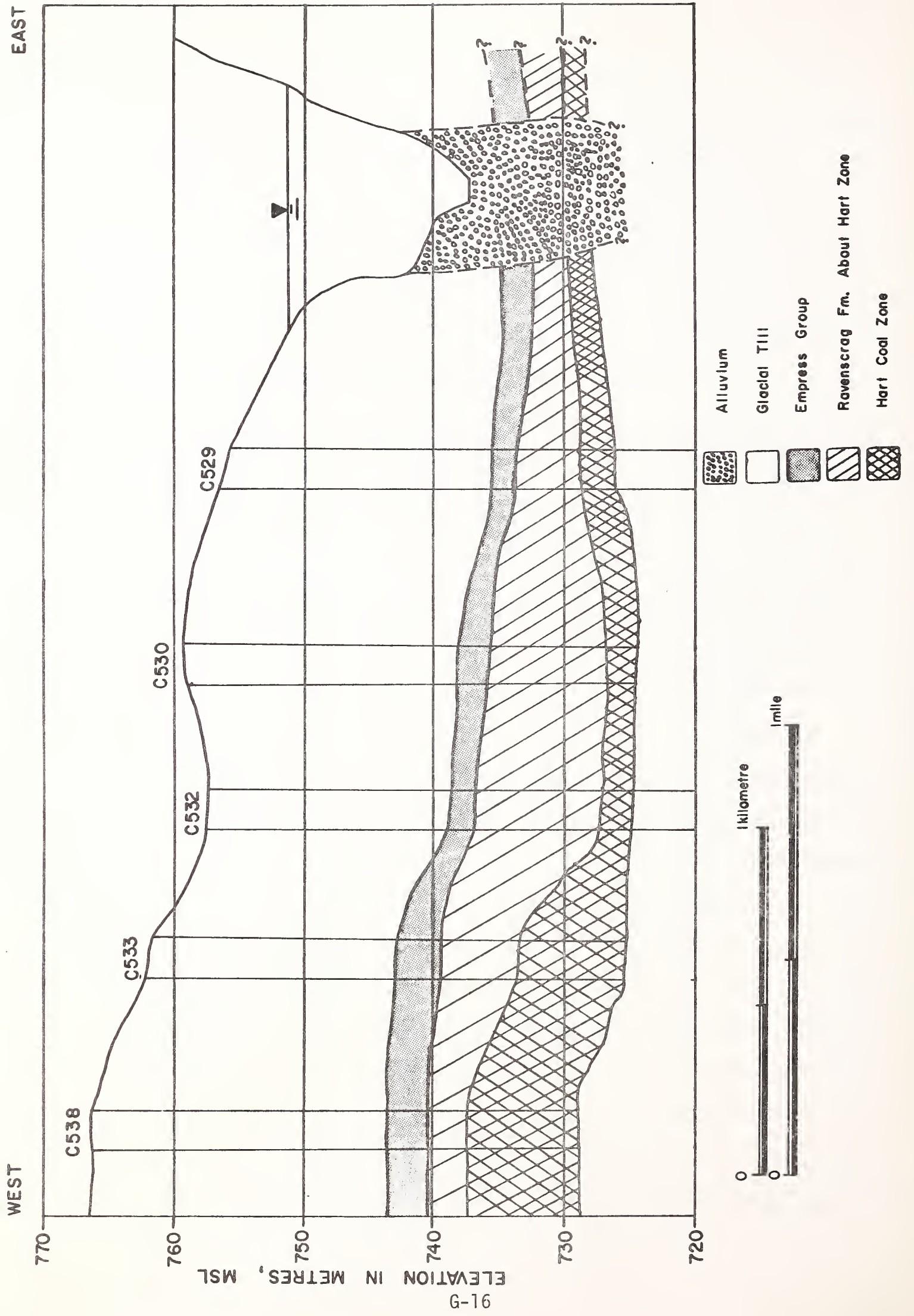


Figure 2. West to East Stratigraphic Cross Section of the Proposed Ash-Lagoon Location.

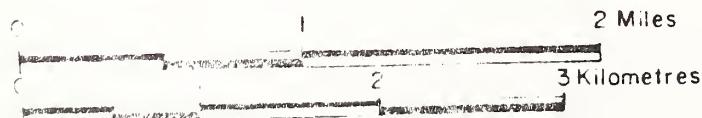
### The Ground-Water Flow Regime

The flow of ground water in the study area is generally controlled by the drainage pattern of the East Poplar River and Girard Creek. The water table in the till, which is recharged directly by local precipitation, is a subdued replica of the surface topography. Ground-water discharges from the till by downward leakage to underlying aquifers; lateral flow to small ponds, marshy areas and the river; and by evapotranspiration. Small ponds and marshy areas which are evident throughout the study area are surface expressions of the water table in the till and probably serve as both recharge and discharge points.

Regional flow conditions in the Frenchman Aquifer, which is the deepest aquifer of concern, are shown in Figure 3. In the immediate area of concern, there is no direct flow of ground water from or to the East Poplar River. The Poplar River does not become a major discharge point for the Frenchman Aquifer until well south in Montana.

In the study area the potentiometric head of the Frenchman Aquifer is several metres higher than that of the Hart Coal Aquifer. Potential, therefore, exists for upward movement of water from the Frenchman Aquifer to the Hart Coal Aquifer. The low hydraulic conductivity of the Lower Ravenscrag Formation materials, which separate the Frenchman and Hart Coal Aquifers, limits the flow between the two aquifers.

-750- Potentiometric Surface Contour, approximate  
where dashed, in metres, msl



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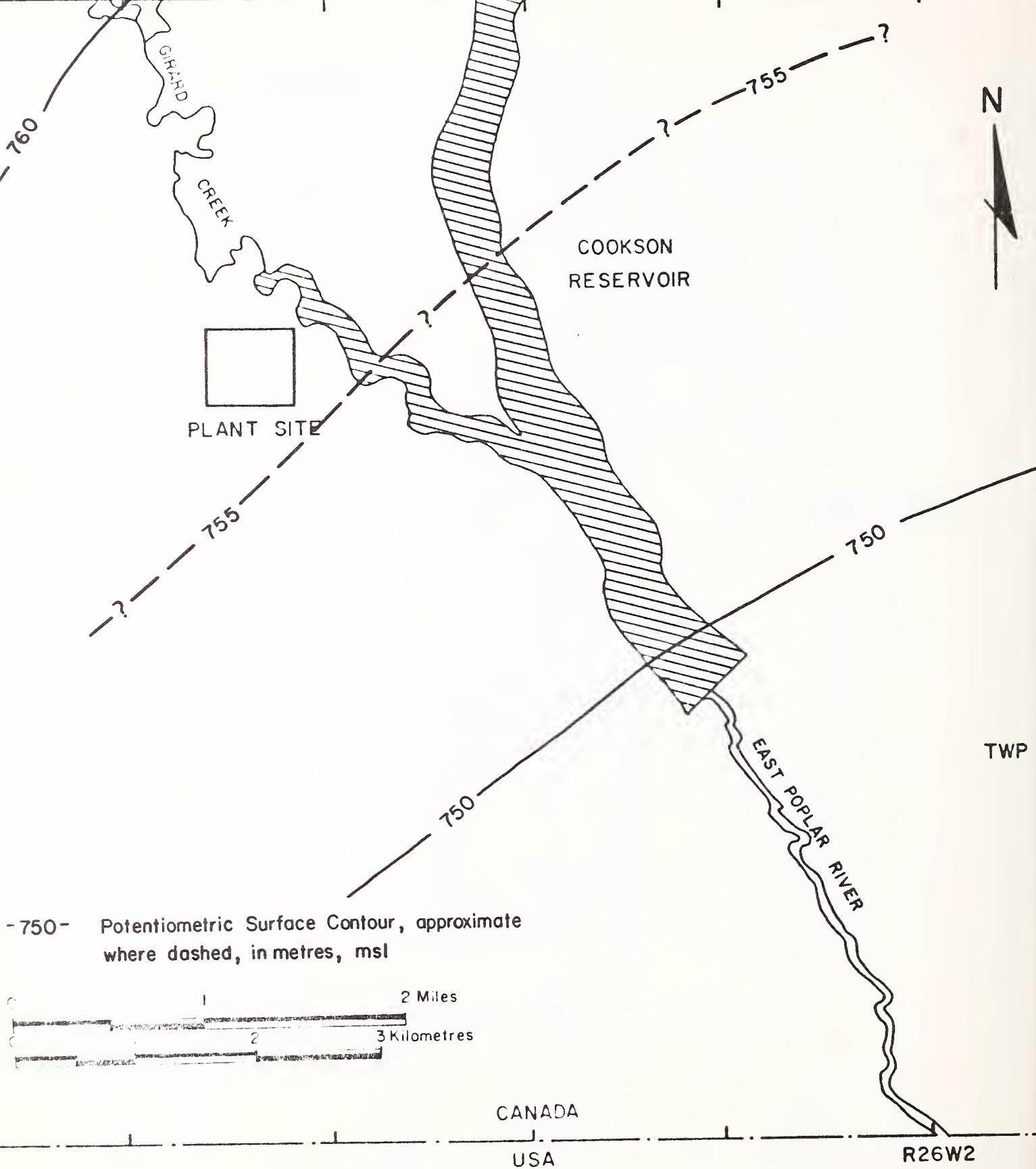


Figure 3. Potentiometric Level of the Frenchman Aquifer,  
Fall 1978 (Whitaker and Vonhof, 1978).

Figure 4 presents the ground-water flow regime in the Hart Coal Aquifer before and after construction of Cookson Reservoir. Prior to reservoir construction, flow was from the aquifer to the East Poplar River and Girard Creek, although the aquifer does not outcrop in the river, but is truncated by the paleochannel of the river (see Figure 2). Ground water discharged from the aquifer into the valley-fill alluvium and then to the river.

The filling of the reservoir has caused a major change in the ground-water flow regime in the Hart Coal Aquifer in the vicinity of the reservoir. Figure 5 shows how water levels in an observation well in the Hart Coal Aquifer have changed due to reservoir filling. Two periods of filling (November 1975 to April 1976 and February 1978 to April 1978) are closely followed by increases of water levels in the aquifer. In the interim time period, water levels in both the reservoir and the observation well stabilized at virtually the same elevation.

No data are available on ground-water flow conditions in the Empress Group prior to construction of the reservoir; however, it appears that flow conditions in both the Empress Group and the Hart Coal Aquifer are much the same. During the second period of reservoir filling, water levels measured in a piezometer in the Empress Group indicate that

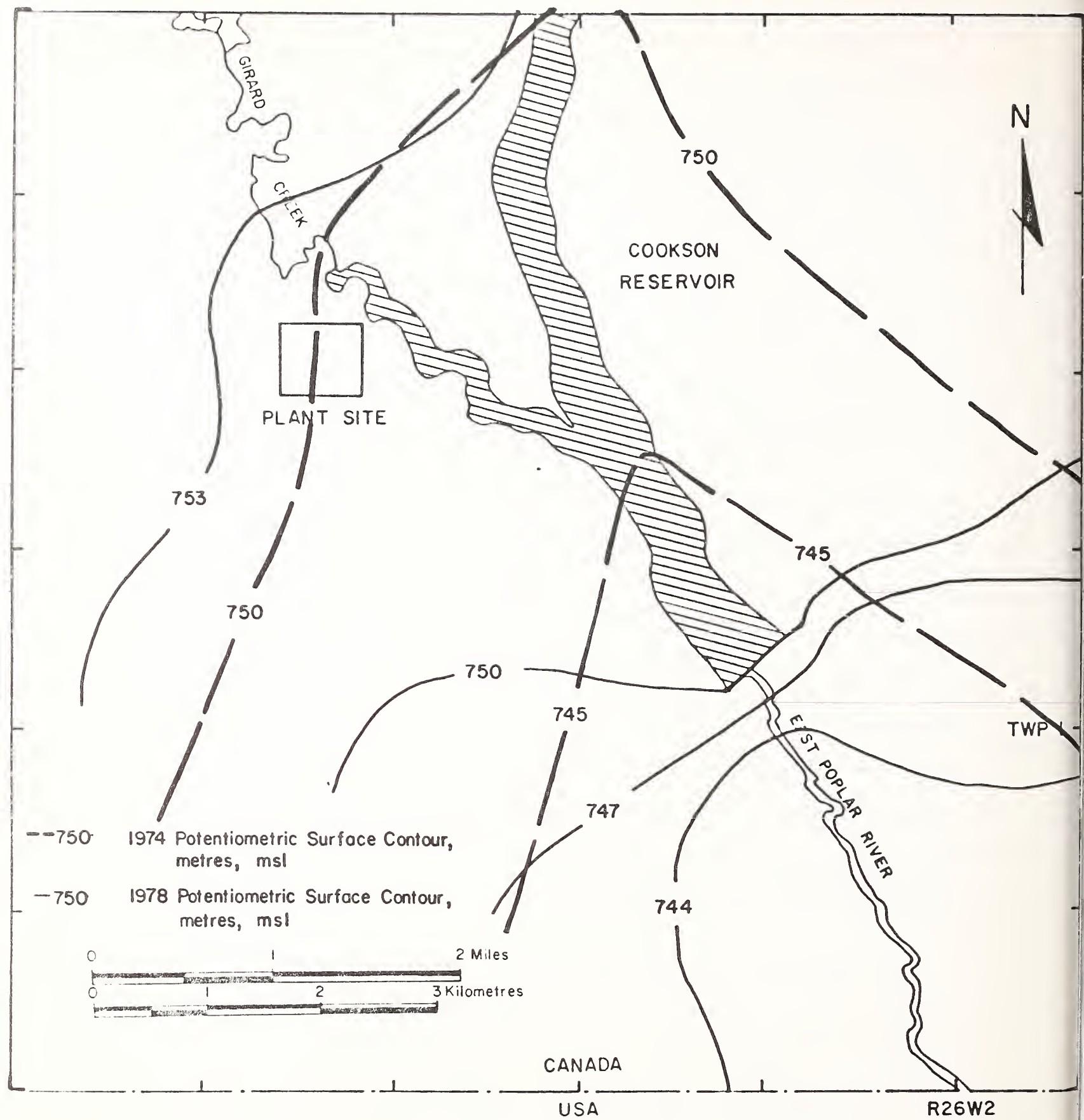


Figure 4. Potentiometric Levels of the Hart Coal Aquifer (Whitaker and Vonhof, 1978).

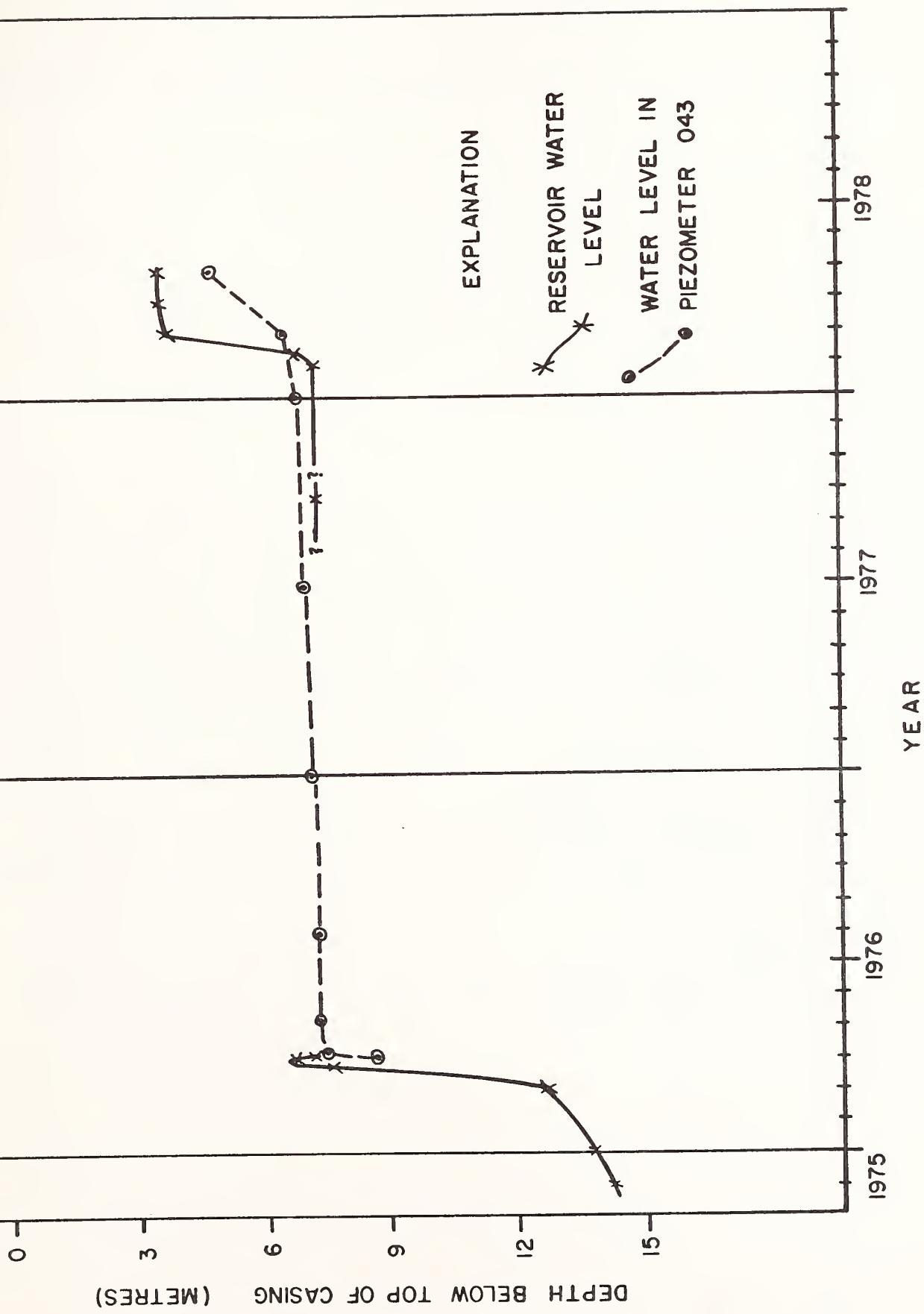


Figure 5. Water-Level Fluctuations in the Hart Coal Aquifer and Cookson Reservoir, November 1975 to June 1978 (Whitaker and Vonhof, 1978).

water-level changes closely coincide with filling of the reservoir (Figure 6). Water levels in both the Empress and Hart Coal Aquifers are now at approximately the same elevation.

The June 1978 and January 1979 ground-water flow conditions of the Empress Group are shown in Figures 7 and 8. The general flow patterns shown should not change greatly with fluctuations of reservoir level, although conditions in both June and January are probably transient prior to final filling of the reservoir.

The ground-water flow pattern around the reservoir is very complex. North of the proposed lagoon site, ground water flows into the reservoir. South of the lagoon site, the reservoir definitely recharges the ground-water system. Around the lagoons, the exact character of the ground-water flow pattern is questionable. It has been assumed, however, although available data do not provide definitive support, that the reservoir is discharging to the ground-water system in the immediate vicinity of the lagoons as is generally indicated by the piezometer data collected in June 1978 and January 1979.

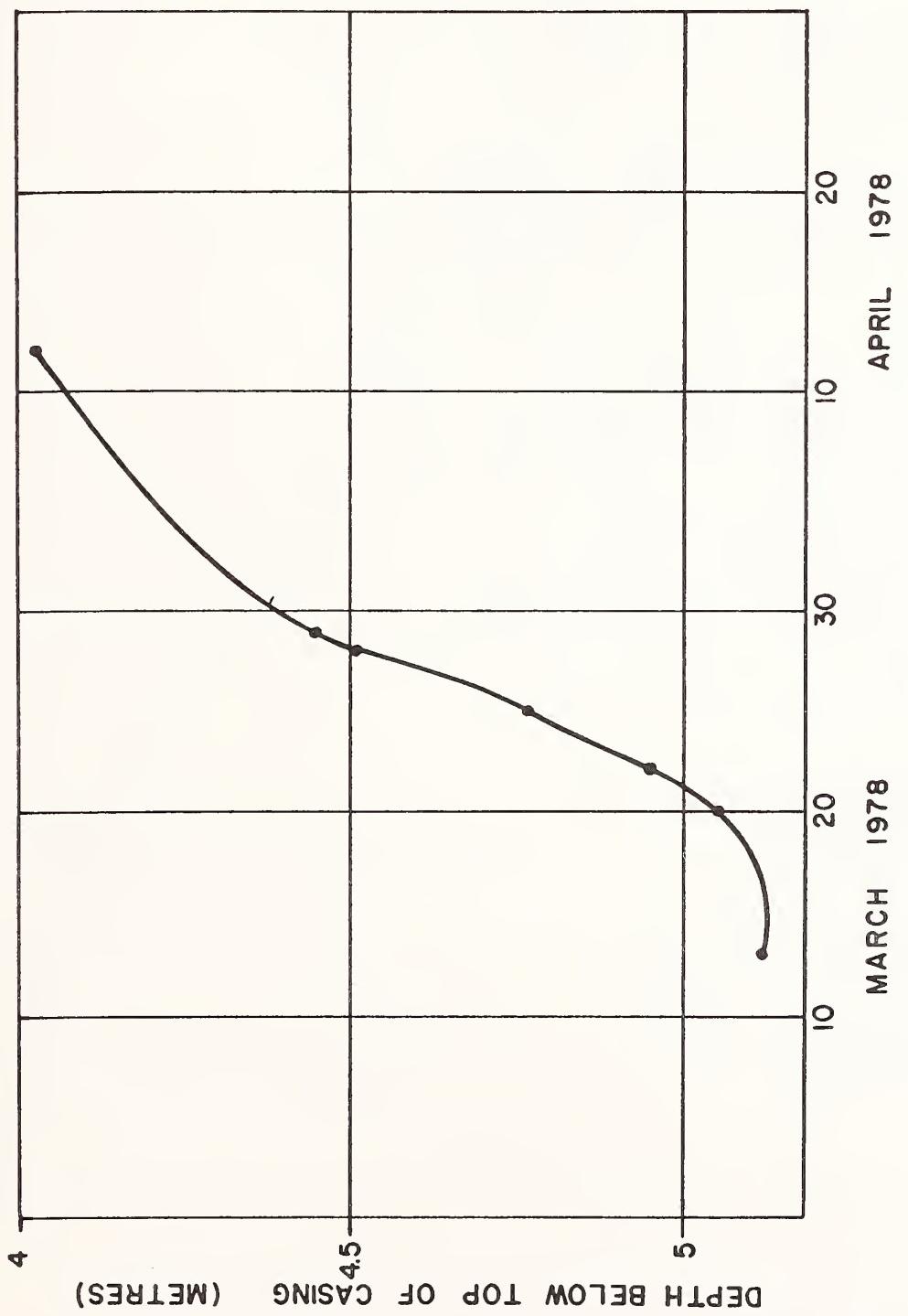


Figure 6. Water-Level Fluctuation in the Empress Group, March 12 to April 12, 1978  
(Ground Engineering, 1978).

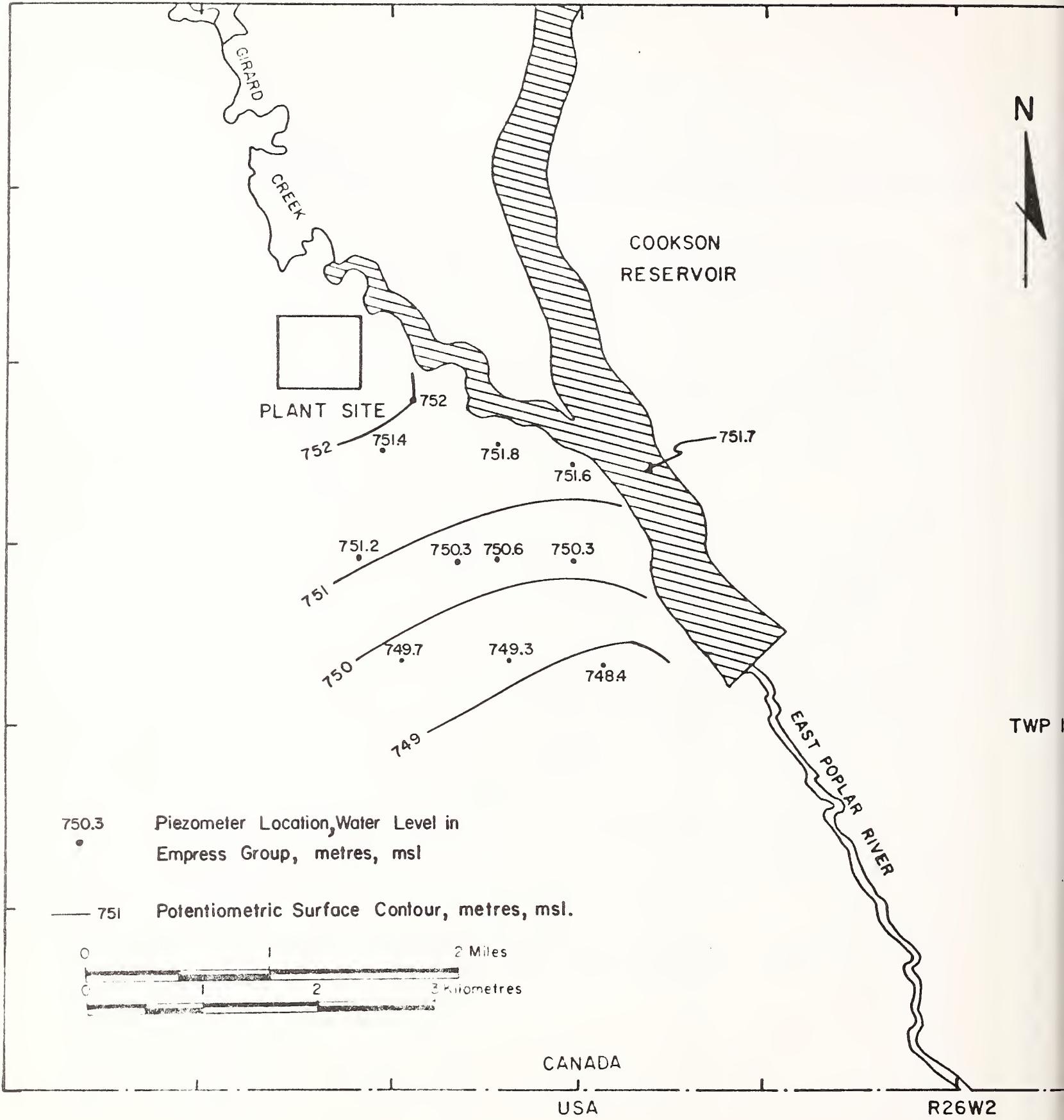


Figure 7. Potentiometric Level of the Empress Group, June 1978.

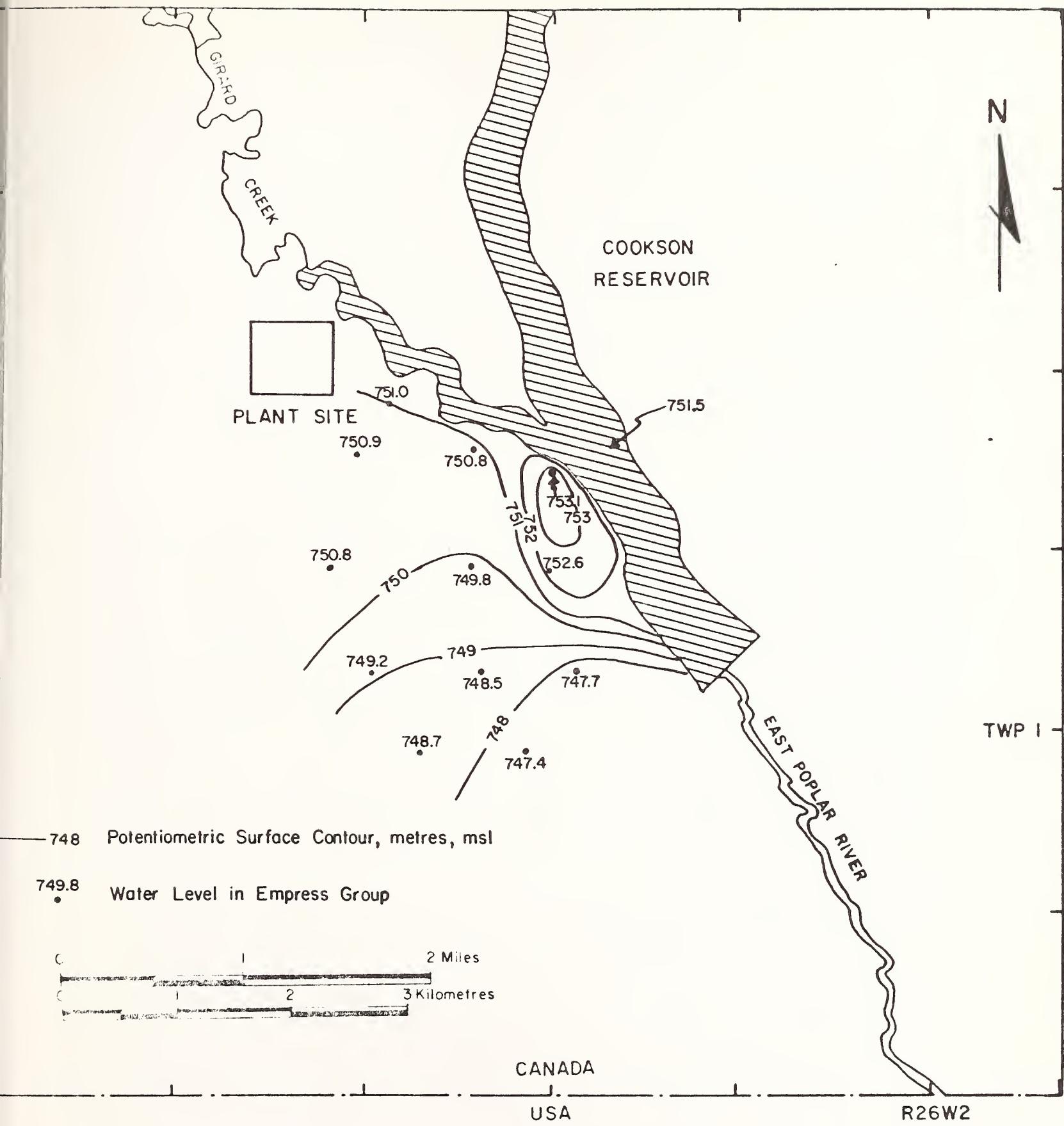


Figure 8. Potentiometric Level of the Empress Group, January 1979.

## ASH-DISPOSAL LAGOON OPERATION

Two ash-disposal lagoon operating schemes have been proposed at the SPC site: (1) once-through liquid flow with settling of ash and release of decant fluid to the reservoir, and (2) continuous recirculation of decant fluid from the lagoons to the precipitators. Each method will be discussed briefly and assumptions regarding size and filling levels will be presented. In each case the lagoons will be receiving approximately 170 l/sec (litres per second) of ash slurry.

The once-through scheme uses three lagoons operating in series. The first lagoon is filled to within one metre of the dike crest; the slurry is then allowed to settle for one month followed by a one-month decanting period of clarified fluid to the reservoir. During the settling and decanting period in the first lagoon, the same sequence of operation begins in the other two lagoons. This sequence of events is repeated every three months in each lagoon. Fluid levels in the lagoons may fluctuate up to five metres during each 3-month period. To simplify this assessment, however, it has been assumed that the long-term average fluid levels in proposed lagoons 1, 2, and 3 will be three metres above the base, and the long-term fluid levels in lagoons 5, 6, and 7 will be two metres above the base. Table 2 presents the actual ranges to be expected for each lagoon.

Table 2. - OPERATING CONDITIONS - ASH LAGOONS

Lagoon	Base Elevation (m)	Full Elevation (m)	Surface Area ( $1,000\text{m}^2$ )	Average Surface Elevation (m) Once-Through Scheme	Depth of Fluid in Storage (m)	Average Surface Elevation (m) Recirculation Scheme	Depth of Fluid in Storage (m)
1	757	763	391	760	3	---	---
2	759	765	402	762	3	---	---
3	758	762	586	761	3	762	4
4	756	762	602	759	3	---	---
5	762	765	599	764	2	---	---
6	768	771	605	770	2	---	---
7	762	765	608	764	2	765	3

The recirculation method of operation utilizes one lagoon in continuous operation. Fluid will enter the lagoon and fill it to within one metre of the dike crest and remain constant at that level. With new input from the power plant, excess fluid will be decanted from the top of the lagoon and be returned to the plant for use in scrubbing and transporting ash. The settling process will be continuous throughout the operation.

It has been assumed, for this investigation, that lagoons 3 and 7 will be operated by the recirculation method. The lagoons will be maintained at a full level producing a fluid depth of 4 and 3 metres, respectively.

## IMPACTS OF ASH-DISPOSAL LAGOONS

Six scenarios have been developed for assessment during this investigation. The scenarios reflect various lagoon locations and operating methods (Table 3). The currently proposed location of the ash-disposal lagoons is shown in Figure 9. Of primary importance in this study are: (1) the amount of seepage to the ground-water system, (2) the most likely pathways for seepage flow from the lagoons to its final destination, and (3) the proportion of seepage that will end up in the reservoir, in the East Poplar River between Morrison Dam and the International Border, and in other sinks such as deeper ground-water aquifers or local surface-water bodies.

### Seepage to the Ground-Water System

Several assumptions concerning the hydraulic conductivity of the till and the hydraulic head driving the leachate into the till must be made before the quantity of seepage from the lagoons can be calculated. Of these two, the hydraulic conductivity assumption is more critical.

Field tests conducted by Ground Engineering indicate that the hydraulic conductivity of the till can be as high as  $9 \times 10^{-5}$  cm/sec. The reports of Whitaker and Vonhof (1978) and the GWQQC (1978) indicate that till hydraulic conductivity may be as low as  $1 \times 10^{-8}$  cm/sec. This wide

Table 3. Ash Lagoon Scenarios

Scenario	Lagoons Involved	Location	Operating Method
1	1, 2, 3	as proposed	once through
2	5, 6, 7	as proposed	once through
3	3	as proposed	recirculation
4	7	as proposed	recirculation
5	1, 2, 3	west of proposed	once through
6	3	west of proposed	recirculation

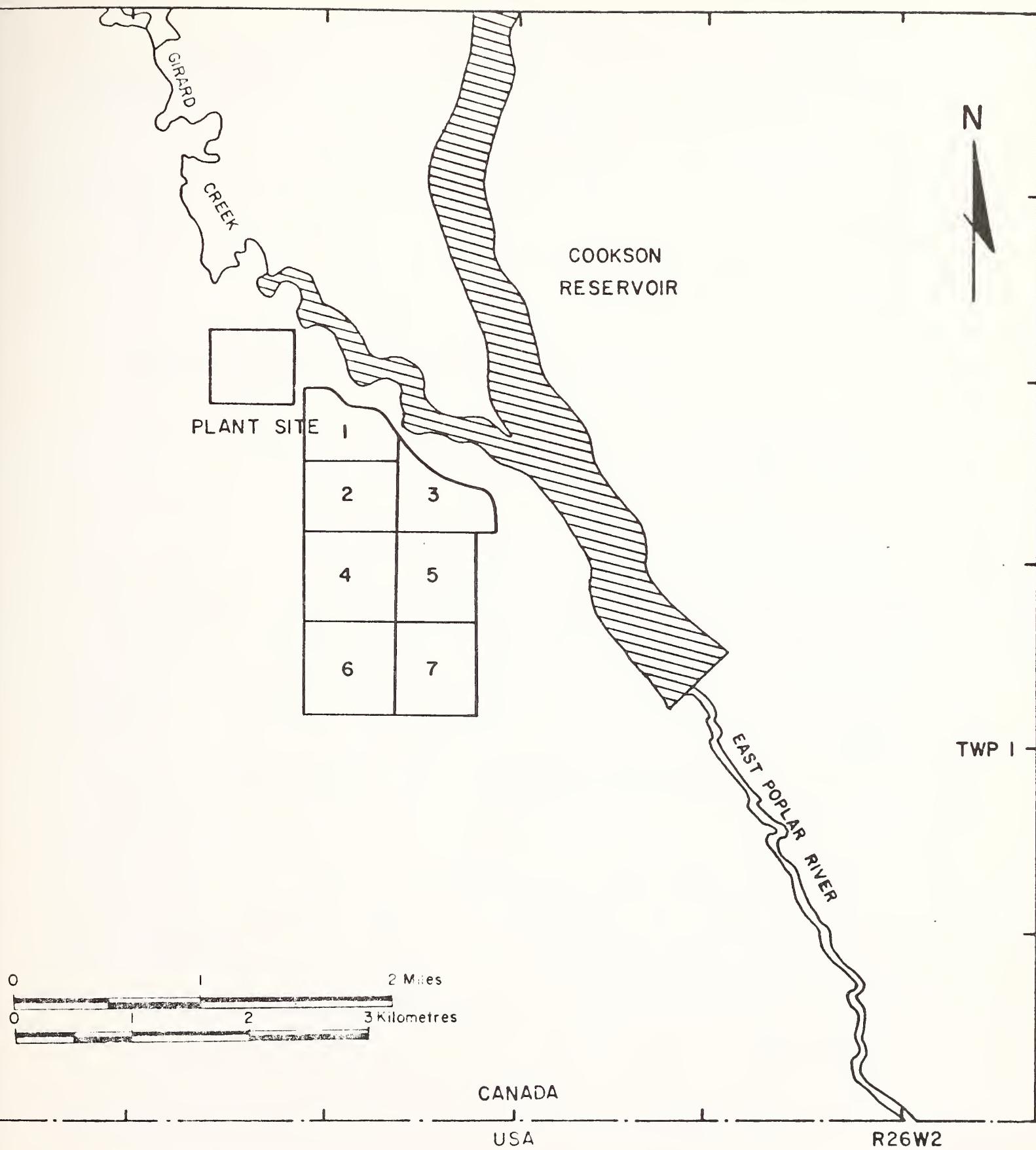


Figure 9. Proposed Ash-Lagoon Location.

range in reported hydraulic conductivity ( $1 \times 10^{-4}$  to  $1 \times 10^{-8}$  cm/sec) essentially means the calculated leachate volumes could range over four-orders of magnitude. For the purposes of this assessment, the following ranges of hydraulic conductivity have been assumed for different till and ash conditions:

Hydraulic conductivity (cm/sec)

Unworked Till	$1 \times 10^{-4}$ to $1 \times 10^{-5}$
Rolled Till	$1 \times 10^{-5}$ to $1 \times 10^{-6}$
Ash	$1 \times 10^{-6}$ to $1 \times 10^{-7}$

The unworked till assumption is based on the observed conditions (Ground Engineering, 1978). The rolled till values are based on a one-order of magnitude decrease with the compaction effort; and the ash assumption is drawn from the literate. It has been suggested that the process of rolling the till will reduce the till's hydraulic conductivity by more than one-order of magnitude; however, pending field testing of the rolling method, we have chosen to use a more conservative value for hydraulic conductivity. These assumptions will be used for all calculations made in this assessment.

Leachate seepage volumes were calculated using the following equation:

$$Q_1 = K \frac{H_W - H_E}{L_t} A \quad (1)$$

where:

$Q_1$  = ash-lagoon leachate seepage ( $m^3/sec$ )

$K$  = hydraulic conductivity ( $m/sec$ )

$H_W$  = elevation of water in lagoons ( $m$ )

$H_E$  = potentiometric level of Empress Group ( $m$ )

$L_t$  = thickness of till ( $m$ )

$A$  = area of lagoons ( $m^2$ )

For this assessment the following hydraulic gradients

$(\frac{H_W - H_E}{L_t})$  have been used:

<u>Scenario</u>	<u>Hydraulic Gradient</u>
1, 5	0.77
2, 4	0.7
3, 6	0.83

As an example, the leachate seepage for Scenario 1, assuming a rolled-till base, is:

$$Q_1 = (1 \times 10^{-7} \text{ m/sec}) (0.77) (1.379 \times 10^6 \text{ m}^2) = 0.106 \text{ m}^3/\text{sec.}$$

Leachate seepage rates for each of the scenarios are presented in Table 4. To generalize, the seepage rate from

Table 4. Leachate Seepage Rates

Scenario	Rate of Leakage ( $m^3/sec$ )		Ash
	Unworked Till	Rolled Till	
1	0.11-1.1	0.011-0.11	0.0011-0.011
2	0.13-1.3	0.013-0.13	0.0013-0.013
3	0.05-0.5	0.005-0.05	0.0005-0.005
4	0.04-0.4	0.004-0.04	0.0004-0.004
5	0.11-1.1	0.011-0.11	0.0011-0.011
6	0.05-0.5	0.005-0.05	0.0005-0.005

three lagoons operating in series with a rolled-till base (the IPWRQB has recommended that SPC rework the till as a liner) will be between 0.012 and 0.12  $\text{m}^3/\text{sec}$ , regardless of their location. As ash builds up in the lagoons, the seepage rate will decrease and could be as low as 0.0012  $\text{m}^3/\text{sec}$ . It is anticipated that the initial leakage rates for the once-through method of operation will be between 0.0012 and 0.012  $\text{m}^3/\text{sec}$ , which may decrease somewhat as the ground-water system becomes saturated. The seepage rate from one lagoon operating by the recirculation method, regardless of location, will be between 0.0075 and 0.075  $\text{m}^3/\text{sec}$ . The rate of seepage will decrease with time due to ash build-up, and may be as low as 0.00075  $\text{m}^3/\text{sec}$ .

#### Seepage Pathways and Destinations

A vertical flow net has been constructed using measured water levels (June 1978) in piezometers completed in the till and Empress Group (Figure 10). The flow net shows that under the currently existing natural conditions (prior to lagoon operation), primary direction of ground-water flow is vertically downward from the till to the Empress Group and laterally in the Empress Group to the south and southeast. Under these conditions, downward leakage from the Empress Group through the Upper Ravenscrag Formation to the Hart Coal Aquifer is expected to be negligible since heads in the Empress Group and the Hart Coal Aquifer are roughly the same.

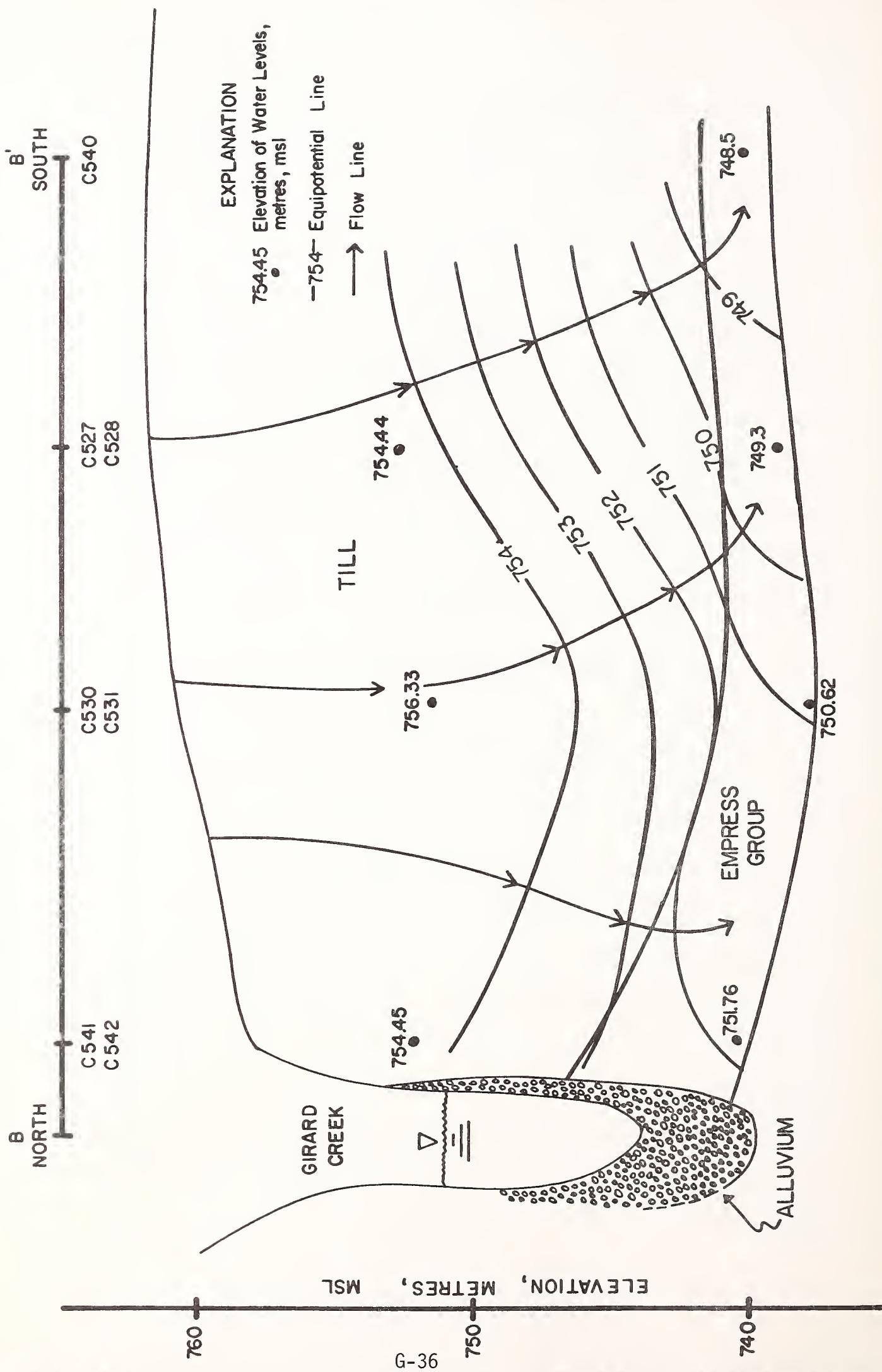


Figure 10. Vertical Flow of Ground Water in the Till and Empress Group, June 1978.

The operation of the ash-disposal lagoons will impose changes on the existing ground-water flow system. It is assumed that such changes will be reflected in both the till and the Empress Group.

A cross section was constructed using a one-to-one vertical and horizontal scale (Figure 11) in an attempt to better understand the impact of lagoon seepage on the ground-water flow system. A north-to-south section was drawn at the point (between lagoons 1 and 3) where the lagoons will be closest to the reservoir. The cross section clearly demonstrates that there would have to be significant lateral flow of leachate through the till for any seepage water to enter the reservoir. It can be argued that since the horizontal hydraulic conductivity of the till is at least ten times greater than its vertical hydraulic conductivity, the tendency for water to flow laterally would be significantly great enough to allow seepage to the reservoir. The hydraulic gradient of the flow system, however, controls actual fluid movement. Under present conditions, the vertical tendency for water to flow between the till and the Empress Group is approximately 200 times greater than the tendency to flow horizontally in the till.

The additional recharge from the lagoons will cause a build up of hydraulic head in both the till and Empress Group. In time, the till will become completely saturated

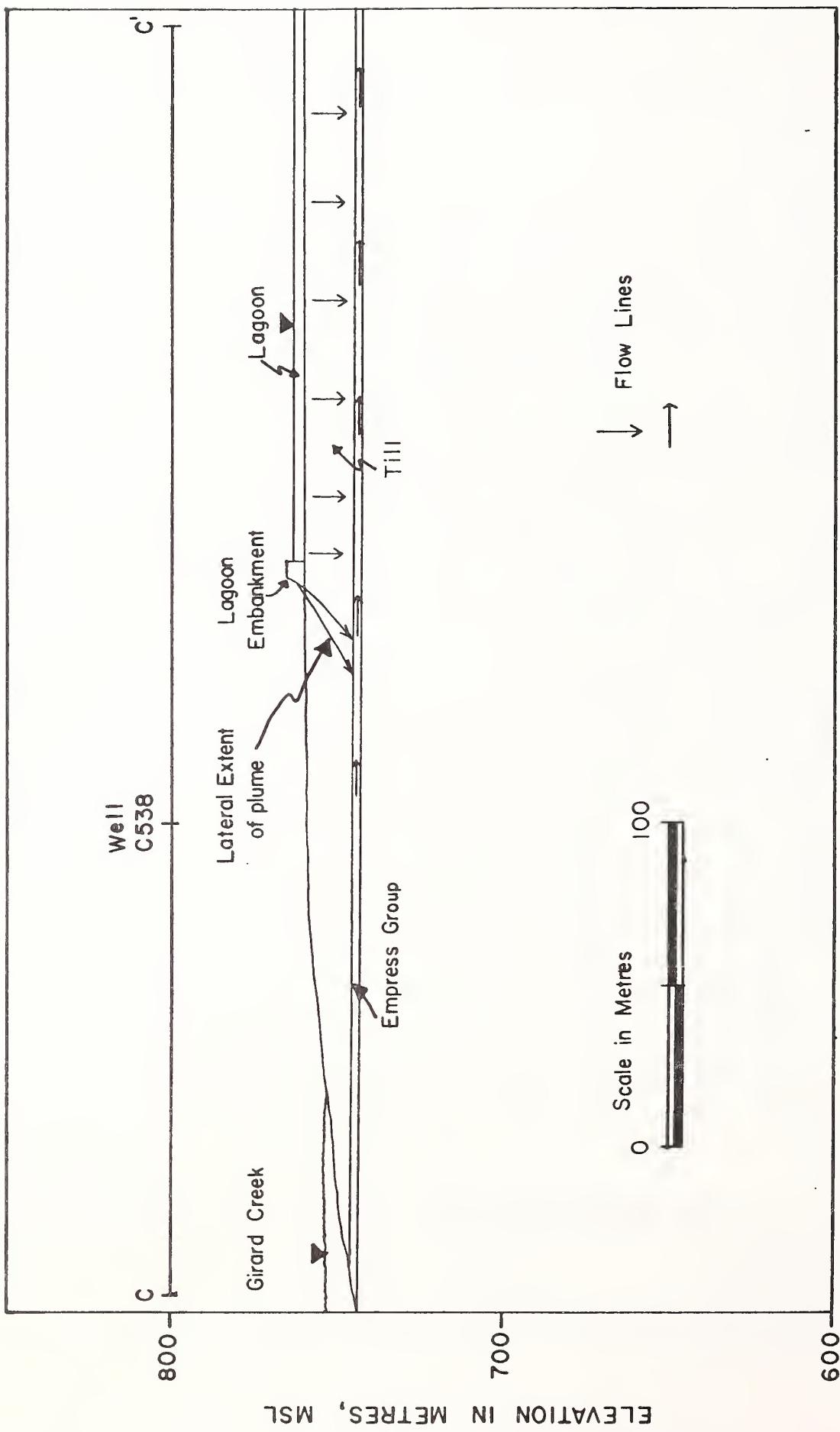


Figure 11. North to South One-to-One Scale Cross Section.

beneath the lagoons and the water table will mound to the land surface. The potentiometric heads will also increase in the Empress Group. Significant head changes in the Empress Group could result in changes in the ground-water flow direction with possible resultant flow to the reservoir.

Due to the short time framework of this study and limited field data, it was not possible to utilize computer modeling techniques to determine the head build up in the Empress Group resulting from lagoon seepage. An analytical model (Hantush, 1967) was used, however, to attempt to characterize the extent of increase in hydraulic head in the Empress Group.

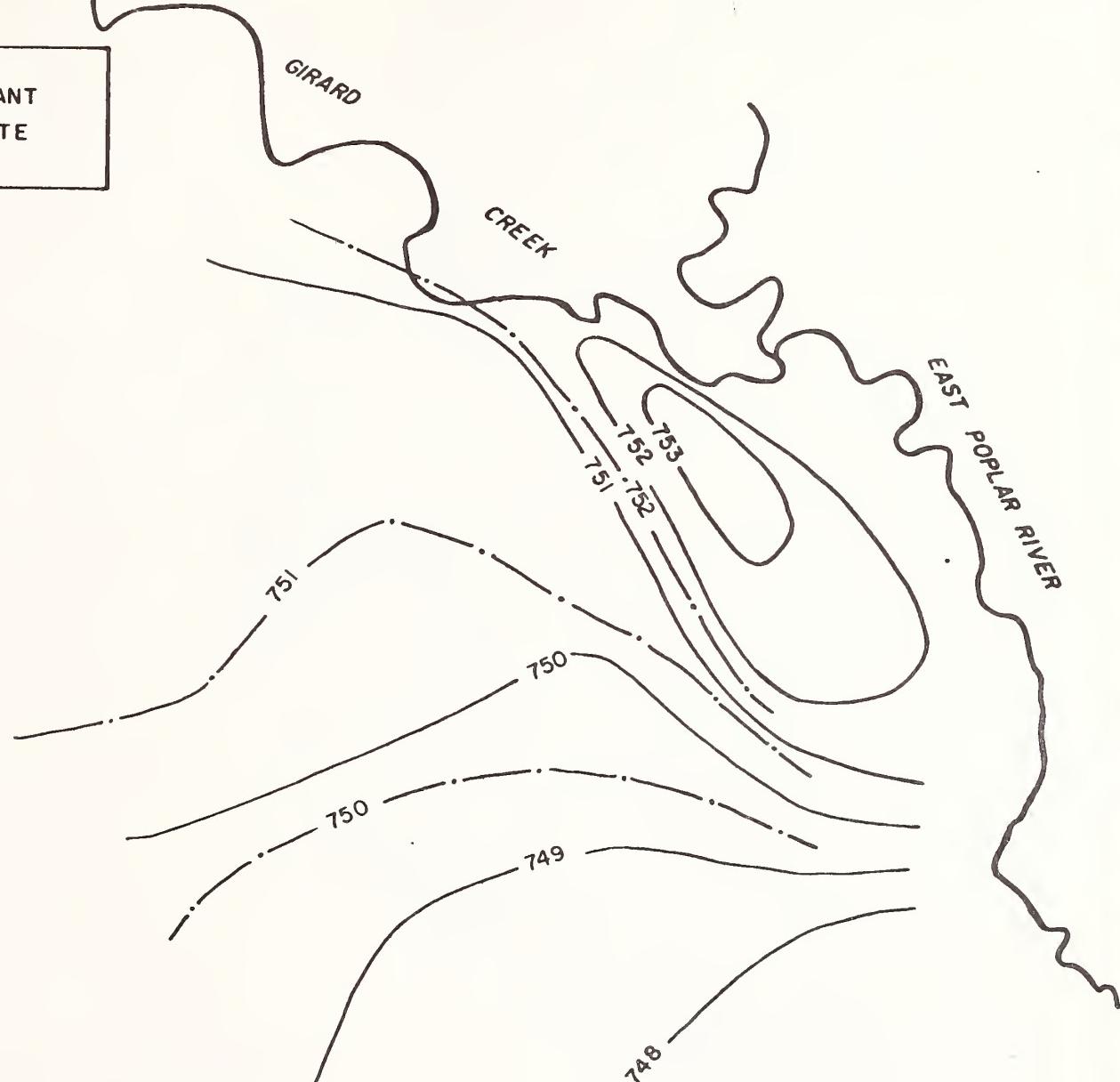
For the calculations, seepage rates of 0.01 and 0.1  $\text{m}^3/\text{sec}$  were used for a circular area equivalent to the area of lagoons 1, 2, and 3. These seepage rates are the approximate rates for rolled-till showing a hydraulic conductivity of  $1 \times 10^{-6}$  and  $1 \times 10^{-5}$   $\text{cm/sec}$ , respectively. The transmissivity of the Empress Group was assumed to be  $200 \text{ m}^2/\text{day}$ . Additionally, the water-level conditions of the Empress Group shown in January 1979 were used, even though they are transient prior to final filling of the reservoir. Changes in hydraulic head were calculated at various distances from the proposed lagoons; these changes were superimposed on the existing potentiometric surface map of the Empress Group and

new potentiometric maps were constructed (Figures 12 and 13). Under conditions presented in Figure 12, the primary ground-water flow direction in the Empress Group would not change enough to permit movement of fluid to the reservoir. Only in the highly unlikely case of large seepage (approximately 10 to 100 times the volume anticipated) reflected in Figure 13 would the gradient in the Empress Group be affected to permit seepage to the reservoir.

A more quantitative approach, such as the use of a computer model, for determining rate and movement of leachate seepage may be desired by the IPRWQB. However, to fully utilize computer modeling capabilities, additional piezometers are needed in both the Empress Group outside of the proposed lagoon area and in the till both at the proposed area and outside of this area. The additional data gathered from these piezometers would enable a more accurate prediction of flow directions and quantities.

Available data and the calculations made to determine the effect of the lagoons on the Empress Group indicate that most of the leachate entering the ground water from the lagoons system will discharge to the surface-water environment in the East Poplar River between Morrison Dam and the International Border. Figures 14 through 19 are highly stylized representations of the probable flow paths of

PLANT SITE



— 750 Potentiometric Surface Contour,  
January 1979 (Metres)

— 751 Potentiometric Surface Contour,  
Showing Impact of Seepage from  
Lagoons

0 1000 Metres

Figure 12. Water-Level Changes in the Empress Group Resulting from  $0.01 \text{ m}^3/\text{sec}$  Leachate Discharge.

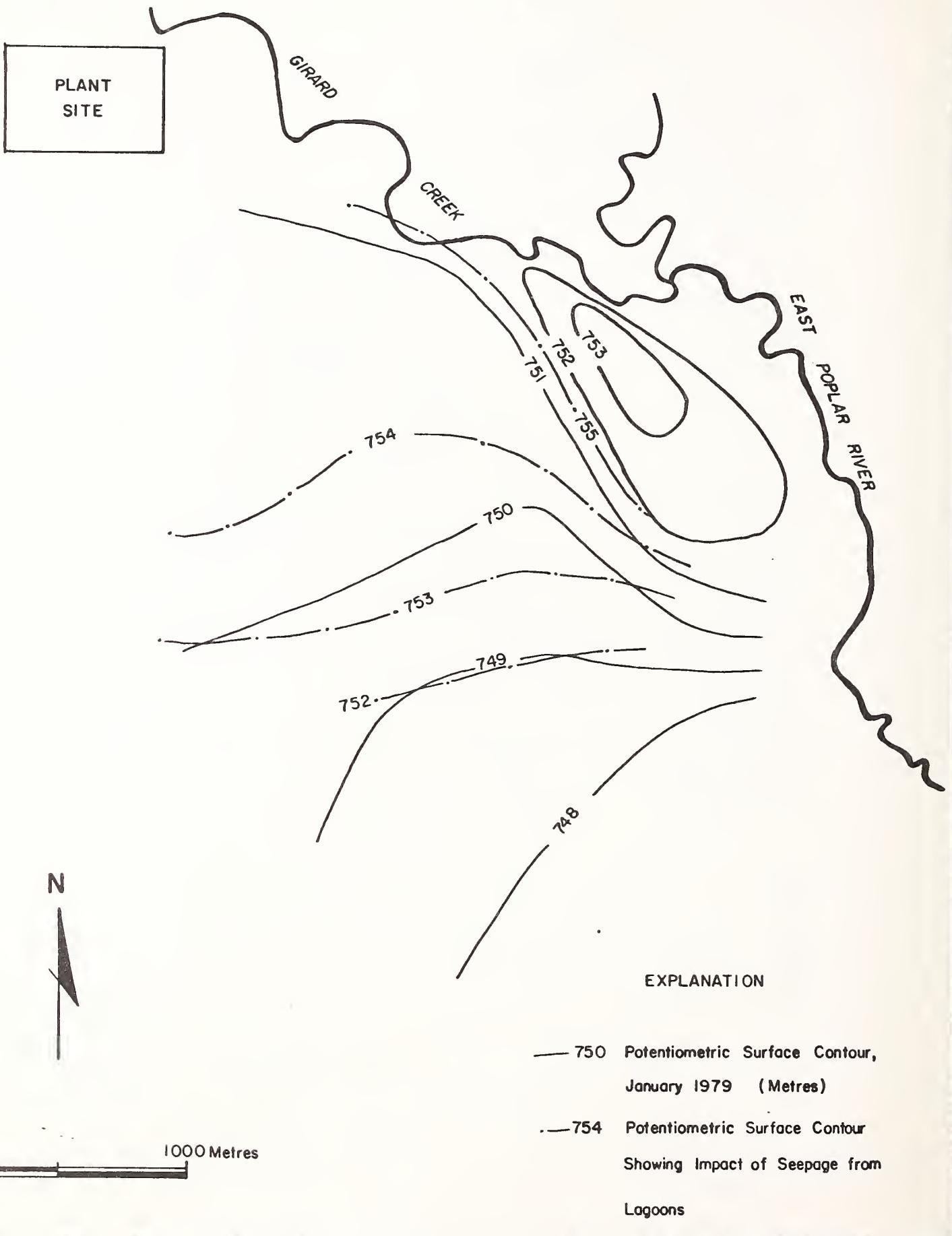


Figure 13. Water-Level Changes in the Empress Group Resulting from  $0.1 \text{ m}^3/\text{sec}$  Leachate Discharge.

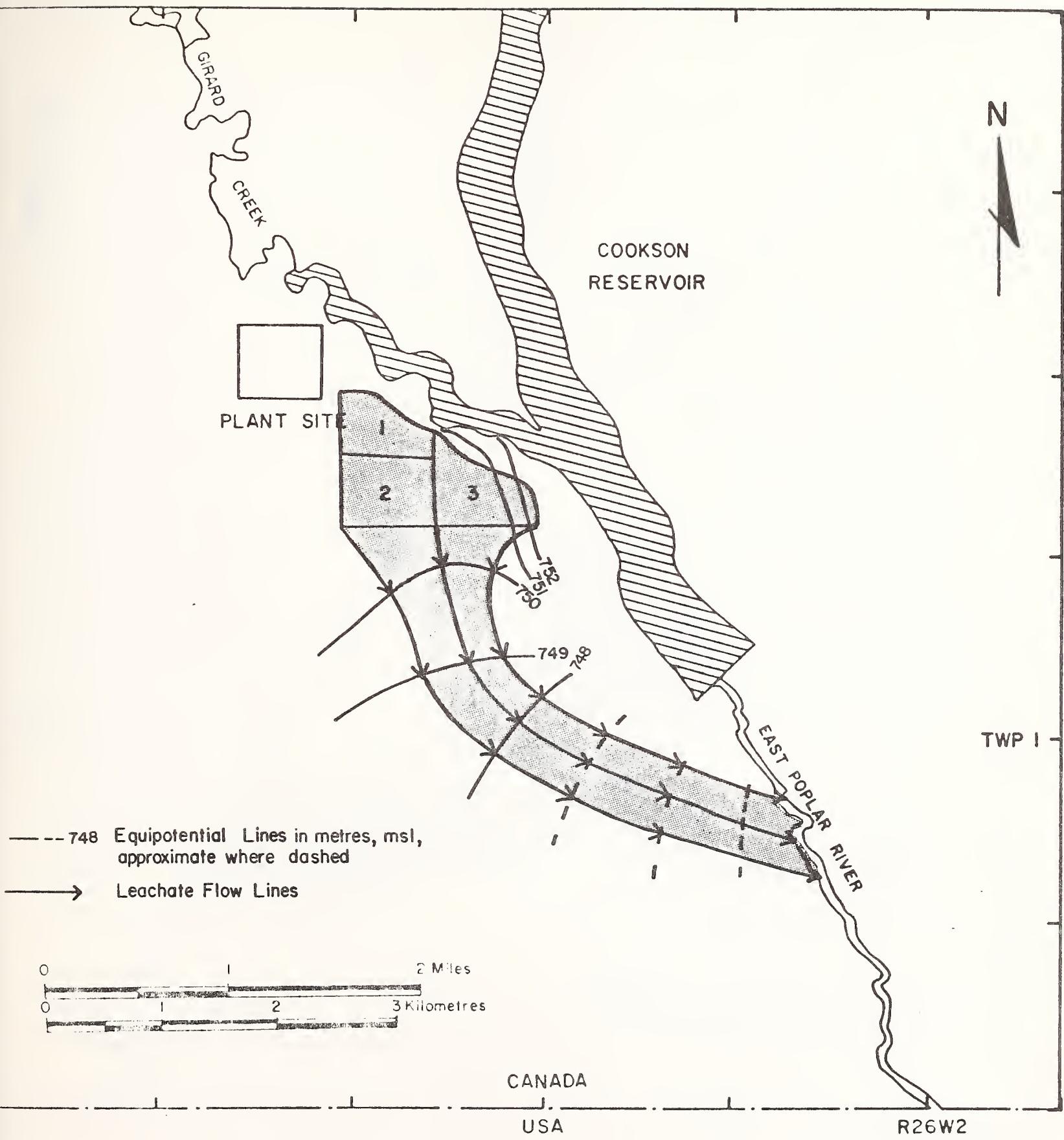


Figure 14. Scenario 1 - Leachate Flow Pattern.

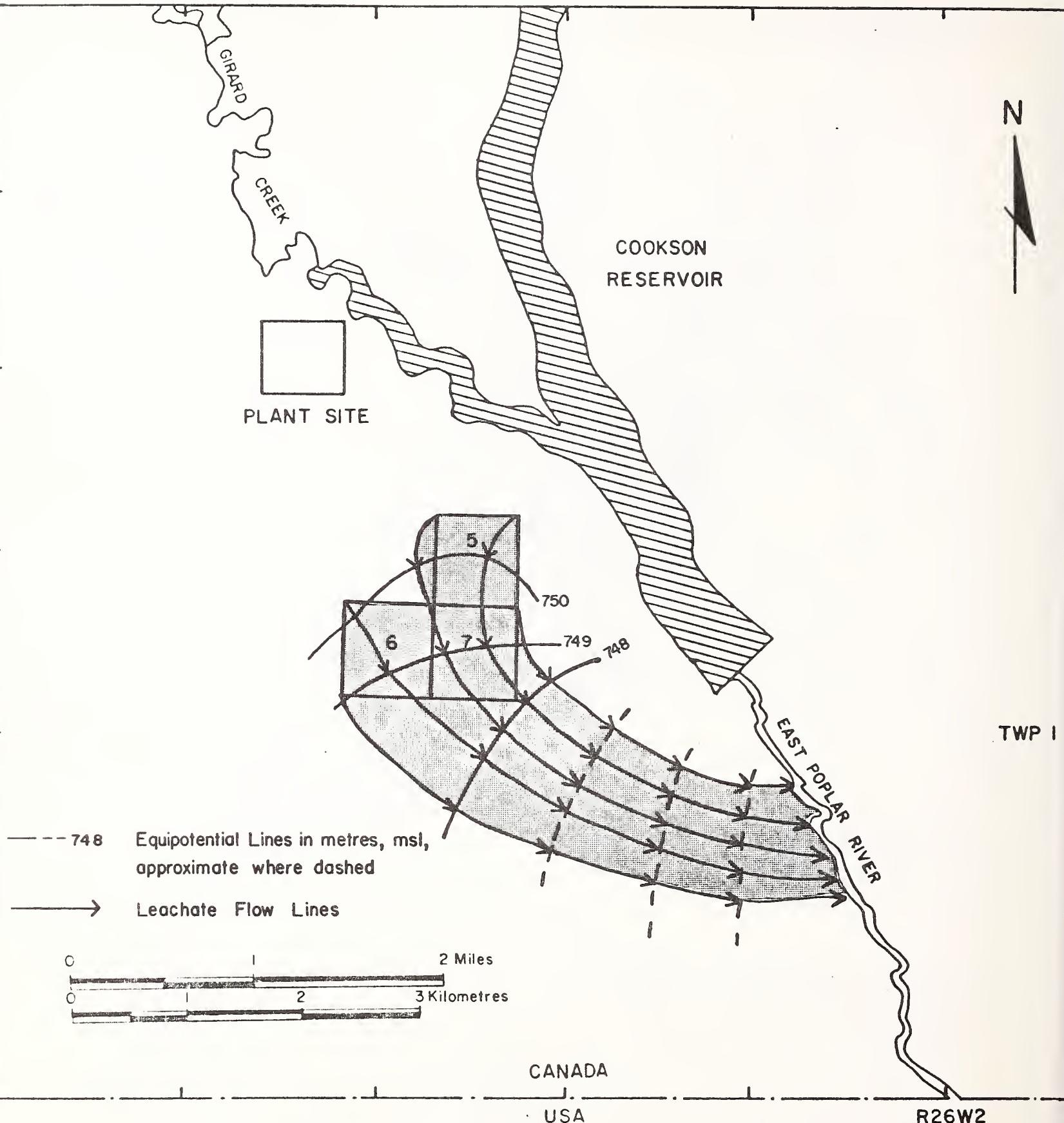


Figure 15. Scenario 2 - Leachate Flow Pattern.

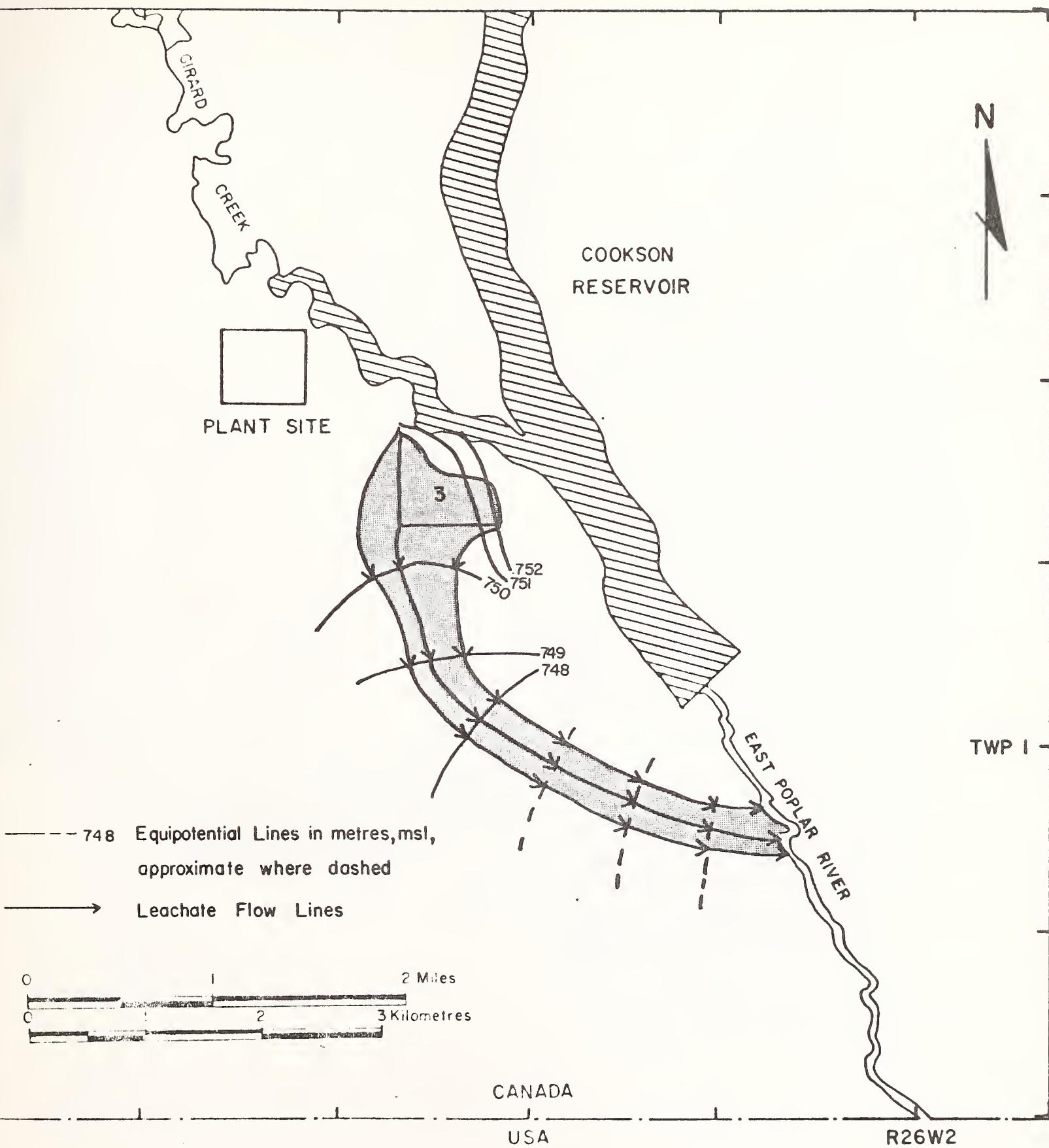


Figure 16. Scenario 3 - Leachate Flow Pattern.

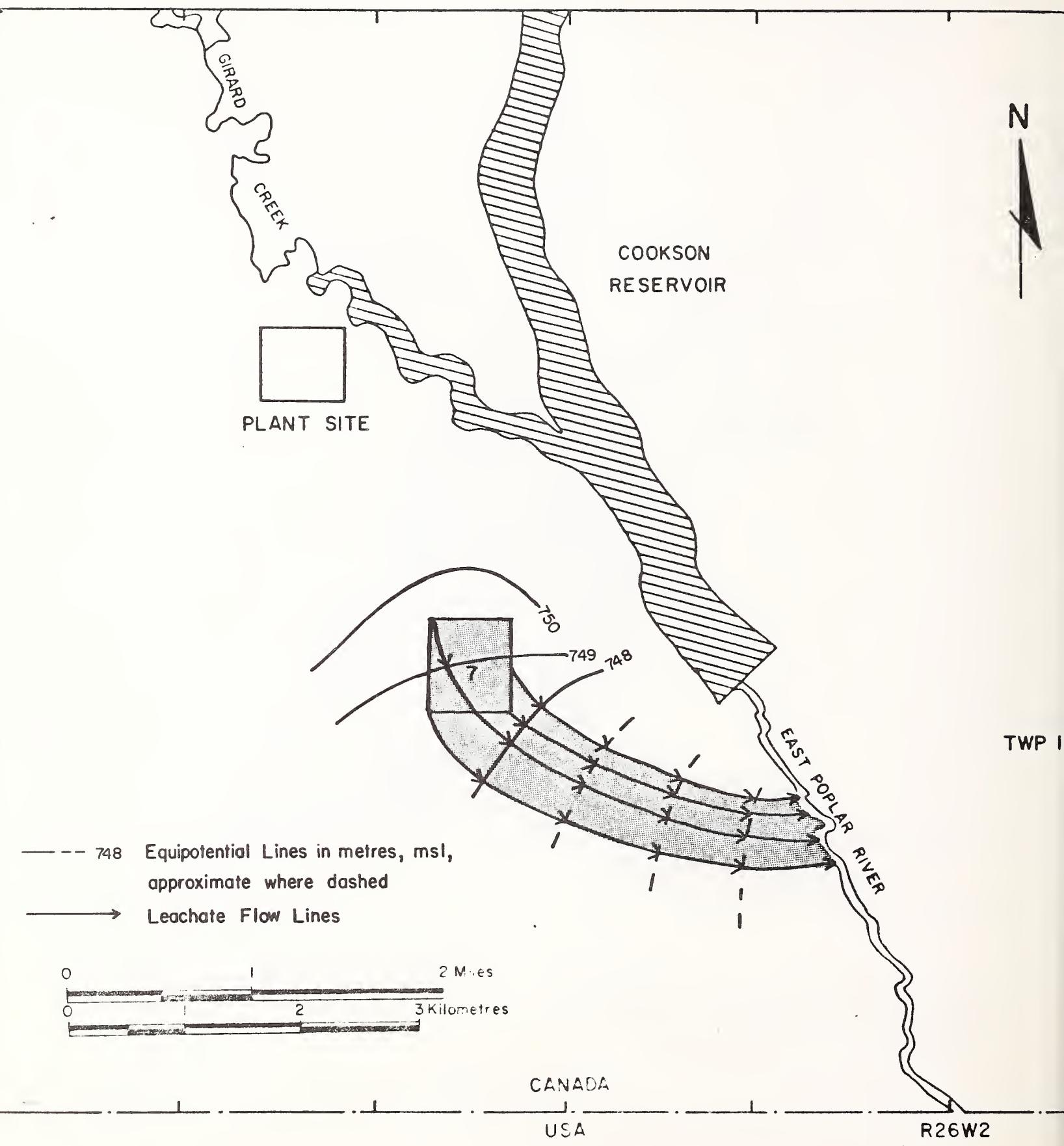


Figure 17. Scenario 4 - Leachate Flow Pattern.

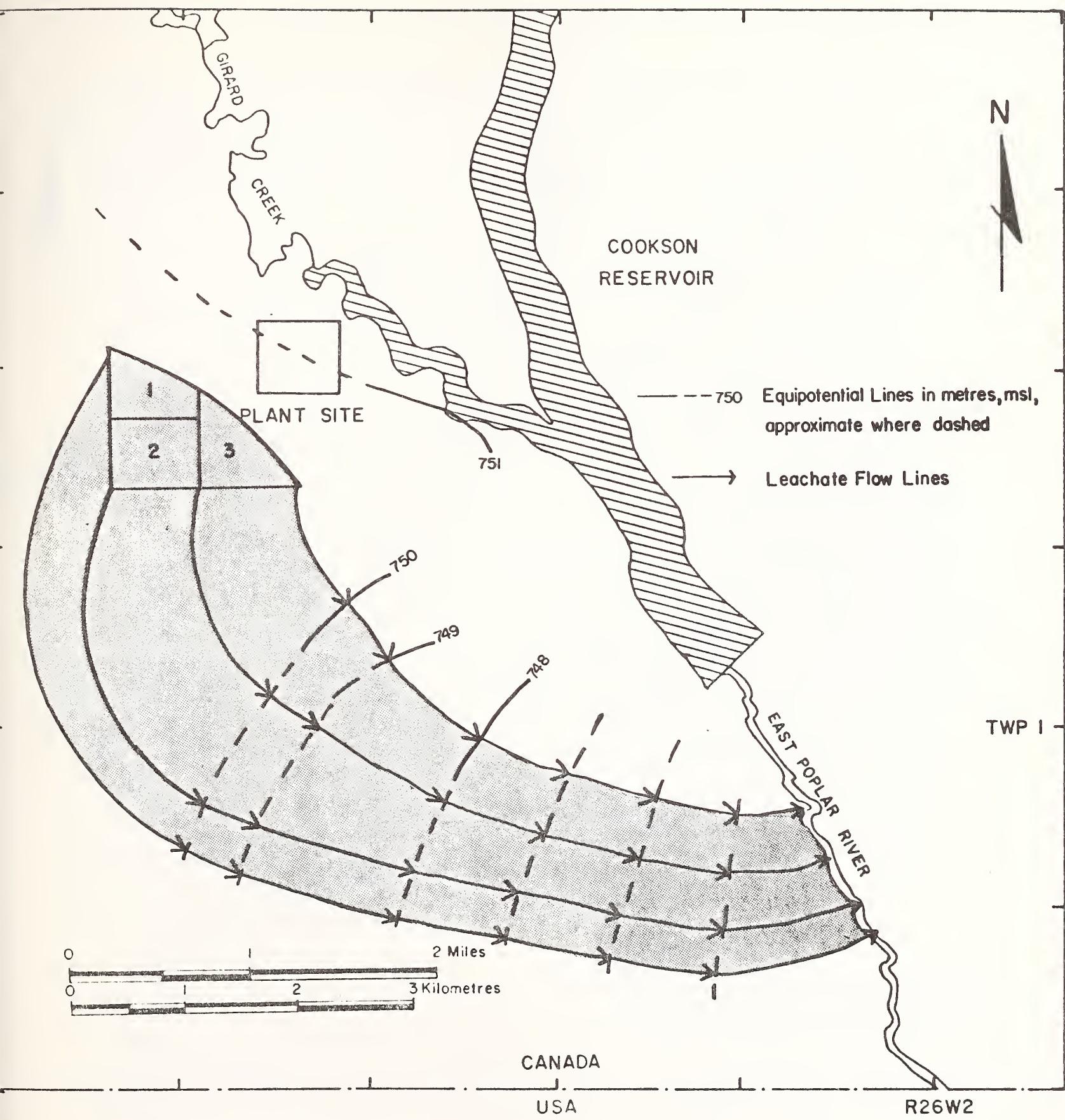


Figure 18. Scenario 5 - Leachate Flow Pattern.

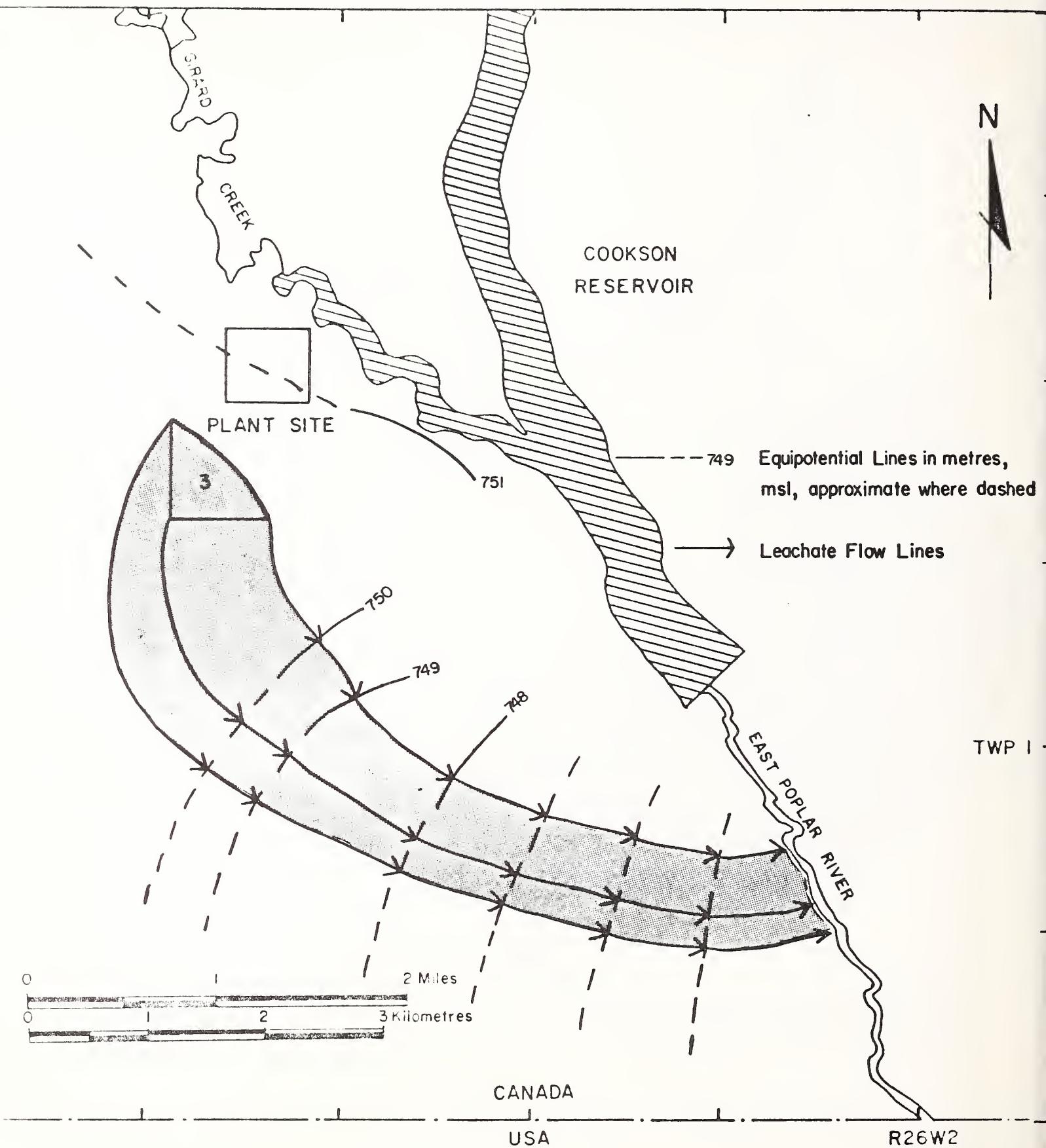


Figure 19. Scenario 6 - Leachate Flow Pattern.

lagoon leachate for each scenario. As previously discussed, it appears unlikely that the development of a ground-water mound would greatly affect this prevailing regional condition.

Because of uncertainties about the structure of the till overlying the Empress Group, there is some indication that a limited amount of leachate could discharge through the till to the cooling-water drainage canal under the operating conditions of scenarios 1 and 3 and to water-table ponds and marshy areas under conditions shown in all scenarios. There is little probability of any movement of leachate to the reservoir via either the till or the Empress Group, as such discharge could only occur if there were reversal of the ground-water flow pattern as a result of mounding beneath the lagoons, which is not anticipated.

The relocation of the lagoons to the west of the presently proposed ash-lagoon location will not affect the ultimate destination of the leachate. The length of time for the leachate to reach the river will, however, be increased (Table 5). The travel-time calculations were made assuming that the leachate will move at the same rate as the water in the Empress Group and that there will be a minimal amount of dispersion and attenuation. Actual travel times may, however, differ somewhat from those calculated.

Table 5. Length of the East Poplar River  
Affected by Leachate

Scenario	Distance from Lagoon to River (km)	Time of Travel (yrs)	Length of River Affected (m)
1	4.8	8.5 - 462	835
2	2.8	5 - 269	635
3	4.8	8.5 - 462	570
4	2.8	5 - 269	424
5	6.0	11 - 577	1,070
6	6.0	11 - 577	715

$$V_E = \frac{KI}{P}$$

$V_E$  = flow velocity through Empress Group

K = hydraulic conductivity of the  
Empress Group

I = hydraulic gradient

P = effective porosity

Based on the theory of continuity, the discharge rate of the plume of contamination to the East Poplar River will be the same as the volume of leachate discharged to the aquifer by the lagoons. The effect on the East Poplar River, however, is related to the size of the discharge plume, thus affecting the dilution potential of the river. The determination of the size of the discharge can be made based on the following equation:

$$Q_E = A_E K_E (dh/dl) \quad (2)$$

where:

$Q_E$  = discharge from the Empress Group ( $m^3/sec$ )

$K_E$  = hydraulic conductivity of the Empress Group ( $m/sec$ )

$dh/dl$  = hydraulic gradient

$A_E$  = cross-sectional area of Empress Group affected by leachate ( $m^2$ )

By rearranging terms, the length of the section of the East Poplar River affected by leachate discharge is determined:

$$L_E = \frac{Q_E}{K_E (dh/dl) T_E} \quad (3)$$

where:

$L_E$  = length of Poplar River receiving discharge (m)

$T_E$  = thickness of the Empress Group (m)

Using Scenario 1 as an example, the length of section of the East Poplar River affected is:

$$L_E = \frac{0.006 \text{ m/sec}}{(1.2 \times 10^{-3} \text{ m/sec})(3 \times 10^{-3}) (2\text{m})} = 833 \text{ m}$$

The length of river affected by the lagoon discharge is calculated to range between 424 and 1,070 m (Table 5).

Observed conditions will be different than that predicted above, and may be expected to be slightly larger, especially if the hydraulic conductivity of the Empress Group is even slightly lower.

Presented in the preceding sections are the calculations made to determine the rates of seepage from the ash-disposal lagoons, the most likely pathways for seepage movement, and the reach of to the East Poplar River affected by seepage discharge. There is a wide range of variability in the results presented due mainly to the range in reported value of hydraulic conductivity.

Since the till will be rolled during lagoon construction and ash build-up will occur during operation, it is reasonable to assume that seepage rates from three lagoons operating in a series will be from 0.0012 to 0.012  $\text{m}^3/\text{sec}$  ( $104$  to  $1,040 \text{ m}^3/\text{day}$ ) (1 to 7 percent of the total effluent discharged to the lagoons). Seepage rates from one lagoon will be considerably less.

Considering the very low leakage rates from the lagoons and the high transmitting capacity of the Empress Group, the most probable destination of virtually all of the leachate will be the East Poplar River south of Morrison Dam and north of the International Border. There may be some minor discharges through the till to the cool-water discharge canal and to ponds and marshy areas. In general, construction on the lagoons south of the power plant will result in the most impact on the East Poplar River. Movement of the lagoons farther west does not diminish the threat and may even increase it slightly.

It can be concluded that approximately one kilometre of the East Poplar River will receive lagoon seepage. This seepage may be expected to affect the river within about 5 years of beginning operation of the lagoons or as late as several hundred years after operation of the lagoons ends.

## CONTAMINATION ABATEMENT ALTERNATIVES

Several methods may be utilized to reduce or control the volume of fluid leaching from the ash lagoons. The most obvious way to reduce the leachate problem along the East Poplar River is to remove the source of the leachate. This may be accomplished by trucking or piping the waste to another location. Secondly, the threat may be reduced by removing the water from the ash prior to its disposal. Dewatering eliminates the entrained media for transporting contaminants to the ground water but does not prevent infiltrated precipitation from transporting contaminants. If it is concluded that the options presented above are not practical in this instance, other abatement programs are available for use.

It has already been suggested by the GWQOC that the lagoons be lined by sealing the till surface with bentonite or by rolling and reworking the till to reduce its hydraulic conductivity. This single liner approach will reduce leaching as shown in the preceding section, but it does not eliminate leaching. Using two liners with a leachate collection system between them should greatly reduce the volume of leachate entering the till. The fluid that passes through the first liner is collected and removed. The collected leachate must then be handled in some environmentally acceptable manner. Liner systems are difficult to use, however, because they cannot be easily monitored to ensure their

integrity. Lagoon size is also important when it comes to installations of liners; these proposed ash lagoons may be beyond the limits of existing technology for liners.

A second type of abatement system is designed on the principle that the leachate will enter the ground-water system. A series of pumping wells or trenches are then installed to intercept the contaminant flow. Because it has been concluded that the leachate will move almost vertically through the till to the Empress Group, trenches would not be practical in this case. Although a trench may intercept some fluid which emanates at the base of the dikes, it cannot collect fluid from the center of the lagoon.

As the leachate is expected to move to the Empress Group and almost totally flow to the southeast, it seems feasible to construct a line of pumping wells across the flow path of the leachate to intercept the leachate flow. The fluid thus collected must be handled in an environmentally safe manner.

Respectfully submitted,

GERAGHTY & MILLER, INC.

*William E. Thompson*

William E. Thompson  
Senior Hydrogeologist

March 30, 1979

REFERENCES

- Ground Engineering, Ltd., 1978, Geotechnical Investigation - Proposed Ash Lagoons: GE-141-1.
- Ground Water Quantity and Quality Committee, 1978, Effects of the Saskatchewan Power Corporation Power Plant on Quantity and Quality of Ground Water in the Poplar River Basin, Saskatchewan-Montana.
- Hantush, M. S., 1967, Growth and Decay of Groundwater Mounds in Response to Uniform Percolation: Water Resources Research, Vol. 3, No. 1.
- Saskmont Engineering, 1978, Poplar River Reservoir Final Environmental Assessment: Vols. 1 and 2.
- Whitaker, S. H., and J. A. Vonhof, 1978, Ground-Water Study - Poplar River Basin in Saskatchewan.

## ANNEX H

### PLANT AND RESERVOIR TECHNICAL DATA

Annex H contains the following:

- (1) Elevation - Area and elevation - storage table for Cookson Reservoir (British units)
- (2) Performance curves for the Poplar River Generating Station cooling water pumps
- (3) Description of Lime Softening Computation Procedure
- (4) Elevation - storage tables for Poplar River ash lagoons
- (5) Two articles on the Sewage Disposal System for Poplar River:
  - (i) Engineering Report for Sewage Disposal, Saskmont Engineering Co. Ltd., March 1976.
  - (ii) Specifications - Sewage Treatment Plant.



## ELEVATION - AREA AND ELEVATION - STORAGE TABLE

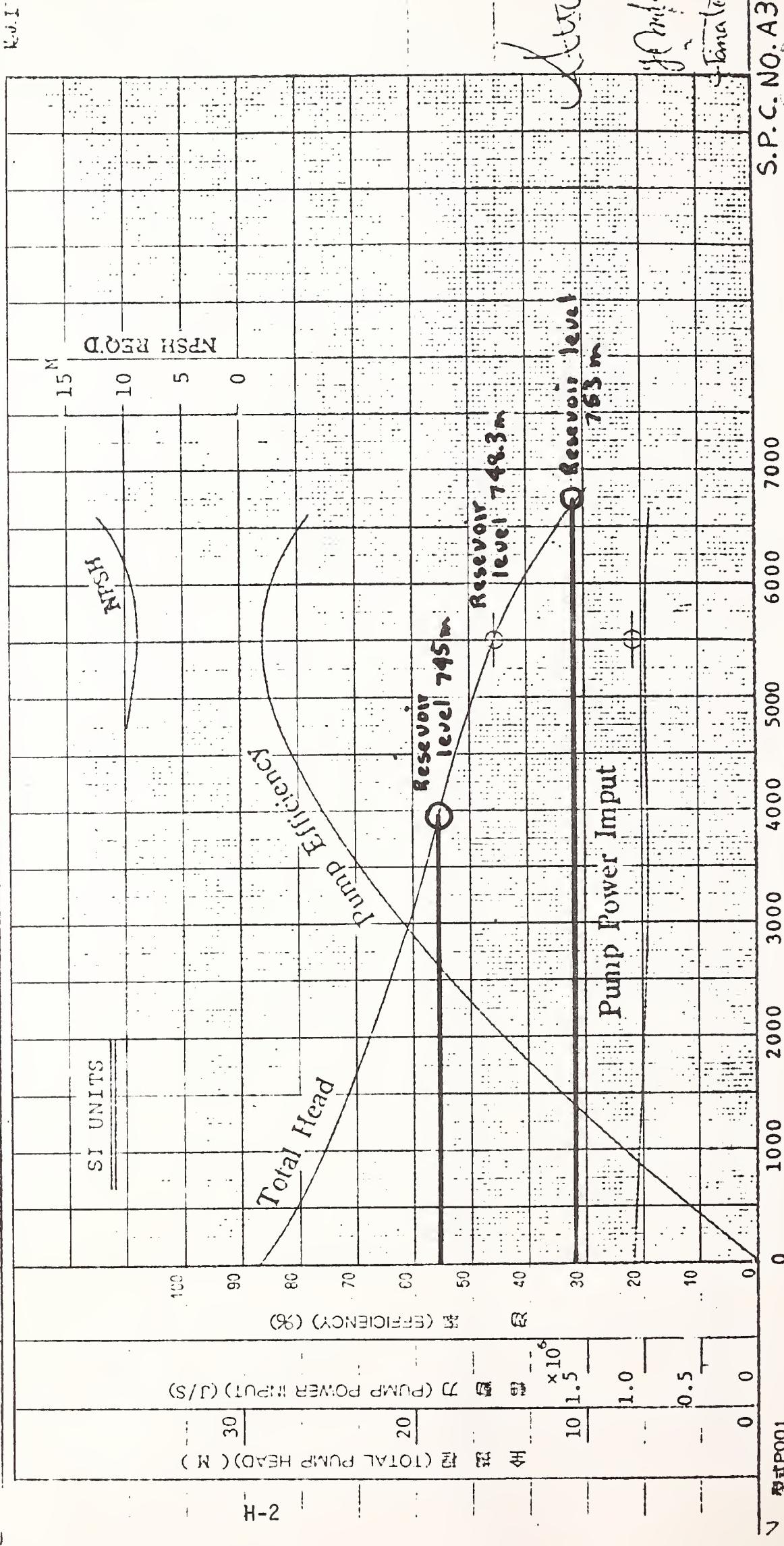
## COOKSON RESERVOIR (BRITISH UNITS)

<u>Elevation (ft)</u>	<u>Area (acres)</u>	<u>Mean Area (acres)</u>	<u>Volume Increment (ac-ft)</u>	<u>Volume (ac-ft)</u>
2425.0 (739.15)	6.2			0
26.0	17.8			12
27.0	29.3	35.1	175	36
28.0	40.9			71
29.0	52.4			117
2430.0 (740.67)	64.			175
31.0	106.			260
32.0	148.	169.5	847	387
33.0	191.			556
34.0	233.			768
2435.0 (742.20)	275.			1022
36.0	309.			1314
37.0	343.	360.0	1800	1640
38.0	377.			2000
39.0	411.			2394
2440.0 (743.72)	445.			2822
41.0	471.			3280
42.0	497.	510.0	2550	3764
43.0	523.			4274
44.0	549.			4810
2445.0 (745.25)	575.			5372
46.0	613.			5966
47.0	651.	670.0	3350	6598
48.0	689.			7268
49.0	727.			7976
2450.0 (746.77)	765.			8722
51.0	795.			9502
52.0	825.	839.5	4198	10312
53.0	854.			11152
54.0	884.			12020
2455.0 (748.29)	914.			12920
56.0	966.			13860
57.0	1017.	1043.2	5215	14852
58.0	1069.			15894
59.0	1120.			16989
2460.0 (749.82)	1172.			18135
61.0	1219.			19330
62.0	1266.	1290.0	6450	20573
63.0	1314.			21863
64.0	1361.			23200
2465.0 (751.34)	1408.			24585
66.0	1484.			26031
67.0	1559.	1597.0	9785	27552
68.0	1635.			29150
69.0	1710.			30822
2470.0 (752.87)	1786.			32570
71.0	1865.			34396
72.0	1944.	1983.5	9918	36300
73.0	2023.			38284
74.0	2102.			40346
2475.0 (754.39)	2181.			42488

性能曲線図(PERFORMANCE CURVES)

COOLING WATER PUMPS - POPLAR RIVER G.S.

製造元 FOR MESSRS	SASKATCHEWAN POWER CORPORATION	日付 DATE Sep. 10. 1976
	POPLAR RIVER POWER STATION	
NO. 5610143	MODEL 1500VZN	TYPE VERTICAL MIXED FLOW PUMP 規定諸目 REQUIREMENTS 5500 L/S 15.3 M



型P001

吐出量 (CAPACITY) ( L/S ) 0 1000 2000 3000 4000 5000 6000 7000

SI UNITS

HEAD (TOTAL PUMP HEAD) (M) 0 10 20 30 40 50 60 70 80 90 100

EFFICIENCY (%) 0 20 40 60 80 100

PUMP POWER INPUT (K/W) 0 10 20 30 40 50 60 70 80 90 100

NPSH REQUIRED (M) 0 5 10 15

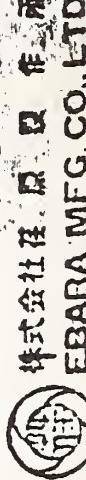
RESERVOIR LEVEL (M) 0 100 200 300 400 500 600 700 800 900 1000

DATE Sep. 10. 1976

Dwg. No.  
EDS-CSPC-004 A

S.P.C. NO. A31  
79-5004

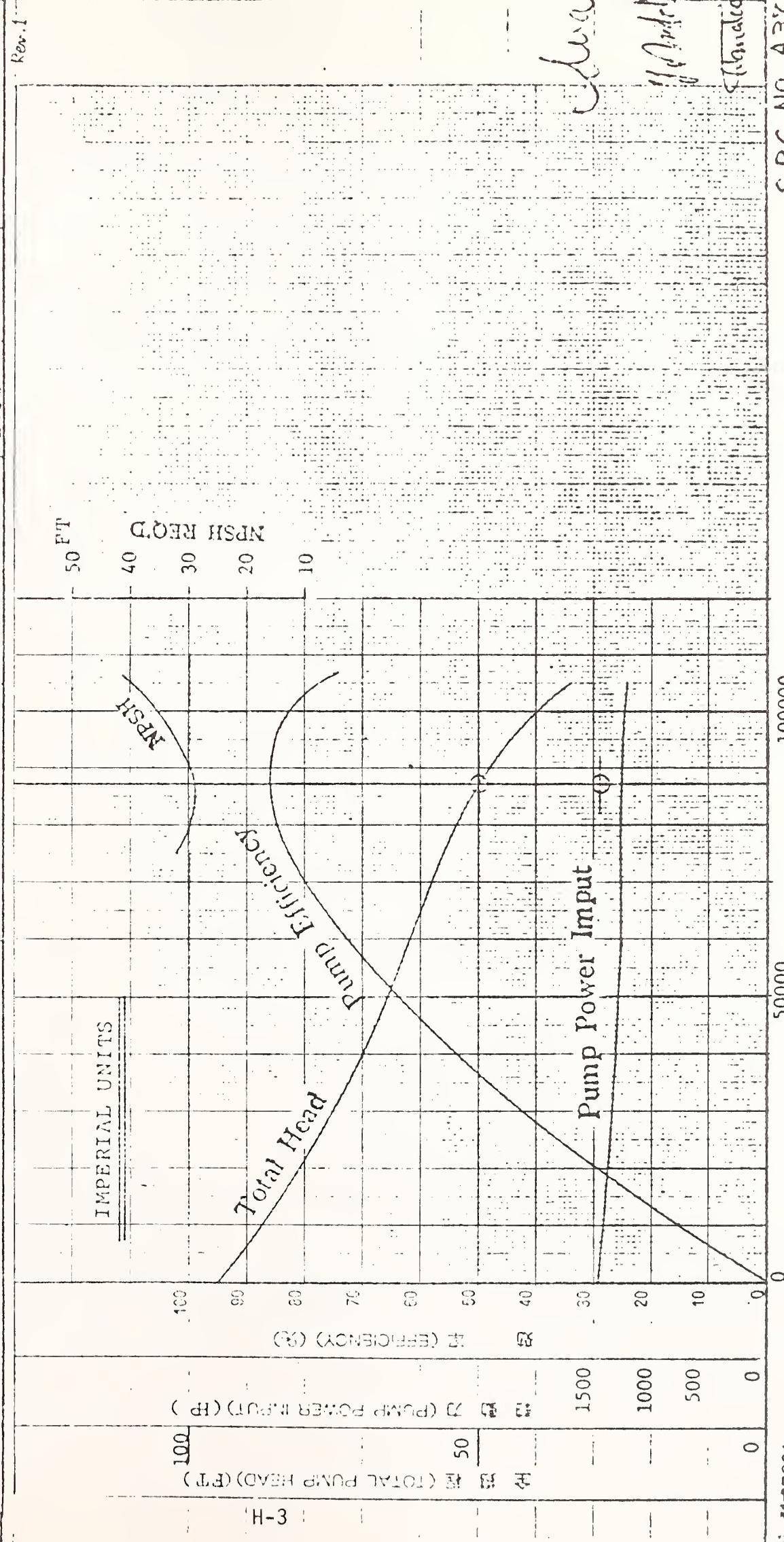
EBARA MFG. CO., LTD.



性能曲線図(PERFORMANCE CURVES)

COOLING WATER PUMPS - POPLAR RIVER G.S.

機種 No. VESSAS	SASKATCHEWAN POWER CORPORATION POPLAR RIVER POWER STATION	用途 SERVICE	循環泵 CIRCULATING WATER PUMP	B 4t
型式 No. MODEL	1500VZRM	流量 USGPM	轉速 RPM	日期 Sep. 10.1976
番号 No. 1610143	規格 REQUIREMENTS	87191	50	H Sp.Gr. = 1.0 Dwg. No. EDS-CSGPC-004B



S.P.C. NO. A372  
179-5004

吐出量 (CAPACITY) ( USGPM )

100000

50000

0

株式会社 旗製作所

EBARA MFG. CO., LTD.

## DESCRIPTION OF LIME SOFTENING COMPUTATION PROCEDURE

(From "Plant Concept - Poplar River Power Station  
Saskmont Engineering Company Limited, 1975)

### (a) Relationship between Total Alkalinity, Calcium Hardness and Ryznar Index

The relationship between pH and total alkalinity provided no carbonates are present is given by the expression:

$$(1) \quad \text{pH} = 4.9 + 1.3 \log. \text{ Total Alkalinity}$$

The saturation  $\text{pH}_s$  is given by the expression:

$$(2) \quad \text{pH}_s = 11.6 - (\log. \text{ Ca.} + \log. \text{ Total Alkalinity})$$

for metal skin temperatures of  $100^{\circ}\text{F}$  and a total dissolved solids content of 500 ppm.

The Ryznar Index is given by the expression:

$$(3) \quad 2 \text{ pH}_s - \text{ph} \text{ and should lie between } 6 - 7 \text{ for scale-free operation.}$$

Substituting 1 and 2 in 3 we get:

$$(A) \quad \log. \text{ Ca} + 1.65 \log. \text{ Total Alkalinity} = 5.65 \text{ (Ryznar} = 7\text{)}$$

$$(B) \quad \log. \text{ Ca} + 1.65 \log. \text{ Total Alkalinity} = 6.15 \text{ (Ryznar} = 6\text{)}$$

Since a Ryznar Index of 7 requires increased blowdown to maintain, equation (B) is the preferred relationship.

It will be undesirable to operate below a pH of 7.0 for reasons of corrosion, and above 8.3 since this requires a high removal of calcium hardness.

From expression (1), the range of total alkalinity for a pH of 7 - 8.3 is 50 - 370 ppm. Total alkalinity in ppm  $\text{CaCO}_3$ , using expression (3), the appropriate range of permissible calcium hardness in the cooling water pond is 2,200 - 80 ppm as ppm  $\text{CaCO}_3$ .

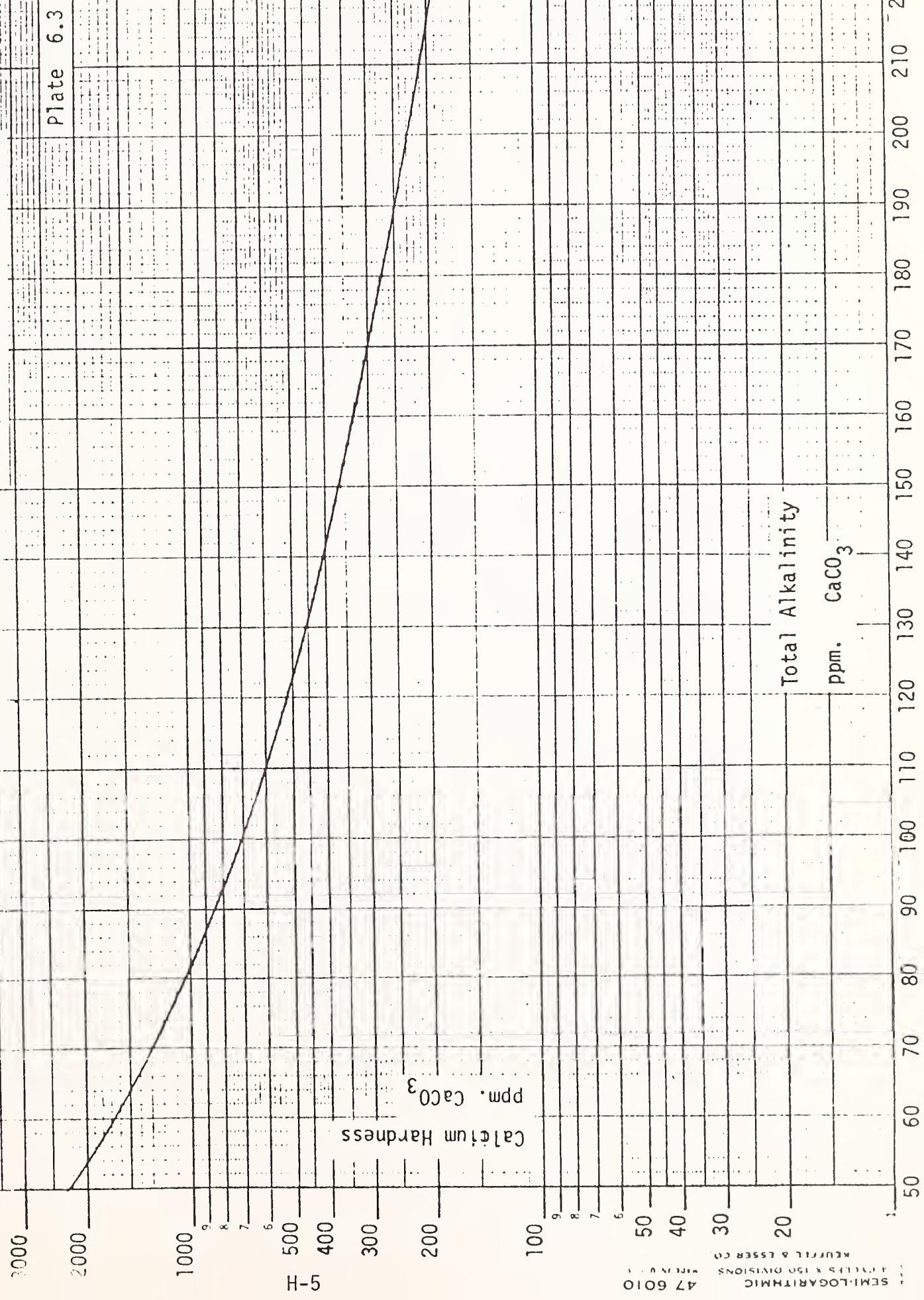
This has been expressed graphically in Drawings 6.3 and 6.4.

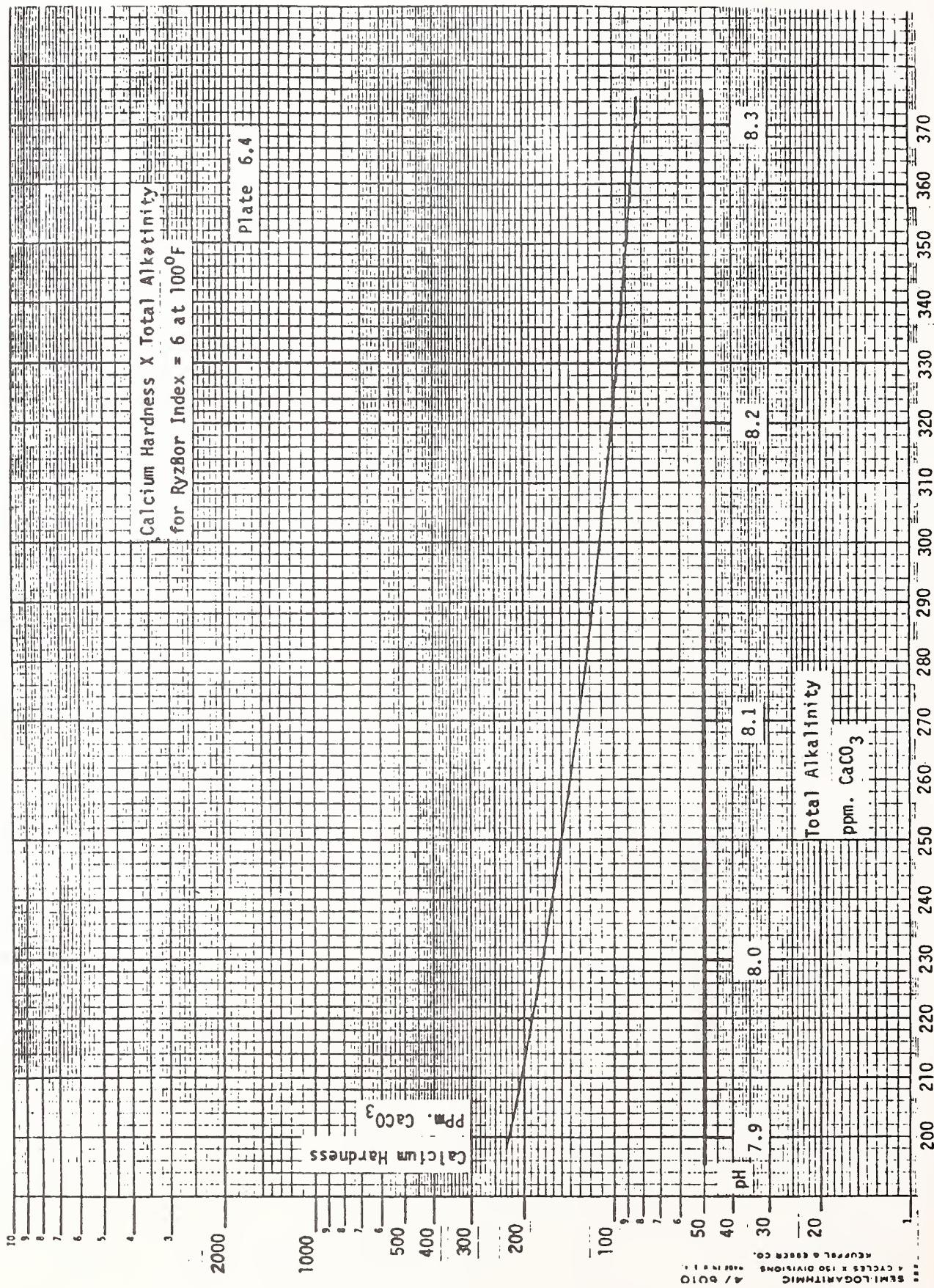
When the calcium concentration in ppm  $\text{CaCO}_3$  is known, the required total alkalinity to maintain a Ryznar Index of 6 may be read from the graph.

It should, however, be stressed that no pH alkalinity should be present if this graph is used.

Calcium Hardness X Total Alkalinity

for Ryznor Index = 6 at 100°F





(b) Basic Equations Used in Reservoir Operational Model

(1) Calcium - Total Alkalinity equation for Reservoir Scale Control

$$\log \text{Ca}_\circ + \log \text{TA}_{\text{lk}} - K_{\text{TDS}} - 5.7375 = \pm 0.001$$

(2) Reverse osmosis feed acidification equation

$$\log 4 \text{Ca}_\circ + 1.5 \log 4 (\text{TA}_{\text{lk}} - \text{Acid}) - K_4 \times \text{TDS} - \frac{1}{2} \log 0.88 \text{ Acid} - 5.009 = \pm 0.001$$

$K_{\text{TDS}}$  for Equations (1) and (2)

$$\begin{aligned} 500 \text{ mg/l} &= 0.171 \\ 1000 \text{ mg/l} &= 0.201 \\ 5000 \text{ mg/l} &= 0.272 \\ 10,000 \text{ mg/l} &= 0.302 \\ 50,000 \text{ mg/l} &= 0.372 \end{aligned}$$

(3) Softening and Stabilization

$$\text{Calcium removal } \text{Ca}_R = \text{Ca} - 35$$

$$\text{Alkalinity removal } \text{TA}_R = \text{Ca} - 35 + 0.2 \text{ Mg} + 25$$

$$\text{TA}_{R1} \text{ per mg/l } \text{Ca}_R = \text{TA}_R / \text{Ca}_R$$

Calcium and total alkalinity are deducted incrementally from analytical data. Initial values calculated from the Calcium - Total Alkalinity equation are within the limits  $\pm 0.001$ . The analysis is then adjusted for calcium, total alkalinity, magnesium and sulphate, total dissolved solids, sum of ions and total hardness are calculated by addition.

(4) Phosphate Reduction

50% removal assumed on passage through softener

$$P = 1 - 0.5 \frac{\text{Ca}_R}{\text{Ca}} \times P_1$$

Ca = Calcium prior to softening

Ca<sub>R</sub> = Total calcium removed by softening

P = Phosphate after softening

P<sub>1</sub> = Phosphate prior to softening

ELEVATION - STORAGE TABLES - POPLAR RIVER ASH LAGOONS

ASH LAGOON #1

<u>Elev (m)</u>	<u>Area (m<sup>2</sup>)</u>	<u>Mean Area (m<sup>2</sup>)</u>	<u>Volume Increment (m<sup>3</sup>)</u>	<u>Volume (m<sup>3</sup>)</u>
753	0	7,500	-7,500	0
754	15,000	42,500	85,000	7,500
756	70,000	115,000	115,000	92,500
757	160,000			207,500
757	391,000	391,000	391,000	
758	391,000	391,000	782,000	598,500
760	391,000	391,000	782,000	1,380,500
762	391,000	391,000	782,000	2,162,500
764	391,000			2,944,500

ASH LAGOON #2

754	0	6,500	13,000	0
756	13,000	76,500	153,000	13,000
758	140,000	200,000	200,000	166,000
759	260,000			366,000
759	402,000	402,000	402,000	
760	402,000	402,000	804,000	768,000
762	402,000	402,000	804,000	1,572,000
764	402,000	402,000	804,000	2,376,000
766.0	402,000	402,000	804,000	3,180,000
766.2	402,000			3,260,400

## ASH LAGOON #3

<u>Elev (m)</u>	<u>Area (m<sup>2</sup>)</u>	<u>Mean Area (m<sup>2</sup>)</u>	<u>Volume Increment (m<sup>3</sup>)</u>	<u>Volume (m<sup>3</sup>)</u>
752	15,000	20,000	40,000	0
754	25,000	45,000	90,000	40,000
756	65,000	182,500	365,000	130,000
758	300,000			495,000
758	586,000	586,000	1,172,000	
760	586,000	586,000	1,172,000	1,667,000
762	586,000	586,000	586,000	2,839,000
763	586,000	586,000	175,800	3,425,000
763.3	586,000			3,600,800

## ASH LAGOON #4

752	0	75,000	150,000	0
754	150,000	220,000	440,000	150,000
756	290,000			590,000
756	602,000	602,000	1,204,000	
758	602,000	602,000	1,204,000	1,794,000
760	602,000	602,000	1,204,000	2,998,000
762	602,000	602,000	602,000	4,202,000
763	602,000	602,000	180,600	4,804,000
763.3	602,000			4,984,600

## ASH LAGOON #5

755	0	5,000	5,000	0
756	10,000	40,000	80,000	5,000
758	70,000	97,500	195,000	85,000
760	125,000	187,500	375,000	280,000
762	250,000			655,000
762	599,000	599,000	1,198,000	
764	599,000	599,000	1,198,000	1,853,000
766	599,000	599,000	119,800	3,051,000
766.2	599,000			3,170,800

ASH LAGOON #6

<u>Elev (m)</u>	<u>Area (m<sup>2</sup>)</u>	<u>Mean Area (m<sup>2</sup>)</u>	<u>Increment (m<sup>3</sup>)</u>	<u>Volume (m<sup>3</sup>)</u>
759	0	22,500	22,500	0
760	45,000	57,500	115,000	22,000
762	70,000	110,000	220,000	137,500
764	150,000	177,500	355,000	357,500
766	205,000	260,000	520,000	712,500
768	315,000			
768	605,000	605,000	1,210,000	1,232,500
770	605,000	605,000	1,210,000	2,442,500
772	605,000			3,652,500

ASH LAGOON #7

757	0	10,000	10,000	0
758	20,000	126,000	252,000	10,000
760	250,000	285,000	570,000	262,000
762	320,000			832,000
762	608,000	608,000	1,216,000	
764	608,000	608,000	1,216,000	2,048,000
766	608,000	608,000	121,600	3,264,000
766.2	608,000			3,385,600

SASKATCHEWAN POWER CORPORATION  
POPLAR RIVER POWER STATION  
ENGINEERING REPORT  
FOR  
SEWAGE DISPOSAL

Saskmont Engineering Company Limited  
810 - 1867 Hamilton Street  
Regina, Saskatchewan

March 1976

## INTRODUCTION

This report on the sewage disposal scheme for the Poplar River Power Station has been prepared for submission to the Saskatchewan Water Resources Commission in accordance with the Water Resources Act.

The Saskatchewan Power Corporation is building a new Thermal Power Station on the East Poplar River near Coronach, Saskatchewan. Site of the Works is located about four miles South of Coronach, Saskatchewan, in Sections 24 and 25 of Range 27, Township 1.

The sewage disposal scheme has been designed on the basis of providing secondary treatment for sanitary waste from the Power Plant, Construction Camp and various maintenance buildings and disposing of the treated wastes in an environmentally acceptable manner.

## 2.0 DESIGN CRITERIA

### 2.1 REQUIREMENTS OF REGULATORY AGENCIES

The following publications have been used as a basis for design:

- A. Canadian Plumbing Code
- B. Saskatchewan Regulation 88/68 Schedule A - Water Pollution Control Regulations
- C. Water Quality Objectives -- Published by Environment Saskatchewan
- D. Sewage Works Design Criteria - Published by Environment Saskatchewan

We have also taken into consideration that the Commission will not permit discharge of effluent into the Poplar River reservoir.

### 2.2 DESIGN LOADS

Based on a peak camp population of 400 and a non-residential power plant operation work force of 300 (working in three shifts) the sewage treatment and disposal scheme will be designed to handle the following estimated loads:

## 2.2 DESIGN LOADS (Cont'd)

- A. Pipelines: hydraulic load from camp = 600 fixture units  
hydraulic load from power plant and site offices = 300 fixture units
- B. Treatment Plant: flow from camp @ 65 gpd per head = 26,000 gallons/day  
flow from plant etc. @ 30 gpd per head = 9,000 gallons/day  
Required Capacity of Plant = 35,000 gallons/day  
BOD<sub>5</sub> production @ 0.17 lb/person/day =  $0.17 \times (400 + 300/3)$   
= 85 lb. BOD<sub>5</sub>/day
- C. Pumps: Peak flow = 200 gallons/minute

## 3.0 SEWAGE COLLECTION TREATMENT & DISPOSAL

### 3.1 COLLECTION

Collection from the Construction Camp will be by means of a 6 inch diameter force main from a pumping station adjacent to the camp. Collection from the Power Plant and other sources will be by means of 6 inch diameter gravity flow sewer pipes with minimum slopes of 3/32 inch per foot. The pipes will discharge into a pumping manhole adjacent to the treatment plant.

### 3.2 TREATMENT

It is intended to use a factory built activated sludge package treatment plant providing secondary treatment. Raw sewage will be pumped in from the collection manhole. Discharge will be into a second pumping manhole adjacent to the plant. Each pumping manhole will incorporate two heavy duty submersible sewage pumps complete with limit switches. A maintenance mechanic will divide his time between this plant and the potable water treatment plant.

### 3.3 DISPOSAL

Treated effluent will be pumped into a storage lagoon with a controlled discharge. Discharged effluent will be used for irrigation. Sludge will be periodically pumped out of the treatment plant and disposed of as landfill. A sludge holding tank will be provided for winter storage.

### 4.0 COST ESTIMATE

The cost estimate for the sewage collection, treatment and disposal system is as follows:

Pipelines and manholes	\$100,000
Treatment plant & pumping stations	\$150,000
Lagoon & sludge tank	<u>\$125,000</u>
<b>TOTAL</b>	<b>\$375,000</b>

### 5.0 SUMMARY

It is our belief that the scheme outlined above for the collection, treatment and disposal of sewage at the Poplar River Power Station is the most effective, economical and environmentally acceptable scheme possible.

Prepared by,

C. L. Payne, P. Eng.

CLP/pap

SPECIFICATIONS

DIVISION 2 - TECHNICAL REQUIREMENTS

SECTION 2.9 - SEWAGE TREATMENT PLANT

2.9.01 DESCRIPTION

A. Definition

Sewage Treatment Plant shall be a factory designed and built activated sludge treatment plant providing secondary treatment of domestic waste. The plant shall include steel tank or tanks complete with all internal partitions, pipework and electrical circuits, and all motors, controls and all other measures and materials required to complete work of this Section.

B. Extent

A complete sewage treatment plant shall be installed in the location shown on the drawings.

C. Related Work Specified in Other Sections

Electrical requirements are specified elsewhere in this Specification.

2.9.02 QUALITY ASSURANCE

A. Requirements of Regulatory Agencies

The final effluent from the sewage treatment plant shall meet the requirements for secondary treatment as listed by Environment Saskatchewan in their booklet "Water Quality Objectives" dated January 1975. Requirements are as follows:

<u>Parameter</u>	<u>% Removal</u>	<u>Effluent Value</u>
1. Ammonia Nitrogen	---	10-20 mg/l
2. BOD (5-day at 20° C)	85-95	15-30 mg/l
3. Coliform Bacteria	99.5-99.9	20.000-200.000 per 100 ml
4. MBAS	80-95	0.2-0.8 mg/l
5. Dissolved Oxygen	---	3-5 mg/l
6. Oil and Grease (Either Solubles)	---	15-25 mg/l
7. pH	---	5.5-10.5
8. Phenolics	---	20-30 ppb
9. Suspended Solids	85-95	15-30 mg/l
10. Phosphorus as PO <sub>4</sub>	---	15-30 mg/l

B. Submittals

The Contractor shall submit the manufacturer's and installation drawings of the equipment in accordance with Clause 1.1.03 of the Project General Requirements.

The Contractor shall submit a maintenance and operating manual in accordance with Clause 1.1.14 of the Project General Requirements.

2.9.03 DESIGN CRITERIA

Flow to the sewage treatment plant will vary as the camp population varies with a maximum flow of 160,000 litres per day (35,000 Imp. gpd) of domestic sewage when the camp is fully occupied and the Power Plant is operational. At present, flow is approximately 50,000 litres per day and ultimately when the construction is completed and the camp is closed, the flow will be 45,000 litres per day.

5-day BOD value shall be assumed to be 240 mg/l.

2.9.04 MATERIALS AND WORKMANSHIP

All materials shall be new and shall be of suitable commercial quality as have been proven successful in their respective uses in similar services and under similar conditions of operations.

2.9.04 MATERIALS AND WORKMANSHIP (Cont'd)

The work shall conform to standards issued by one or more of the following organizations:

- Canadian Standards Association
- American Society of Mechanical Engineers
- American National Standards Institute
- American Society for Testing Materials
- The Canadian Electrical Manufacturers Association
- The National Electrical Manufacturers Association.

The specific standard to be used in each case shall be subject to the approval of the Engineer and shall be the latest edition or revision.

2.9.05 EQUIPMENT

The sewage treatment unit shall have the following features:

- (a) Aeration Tank
- (b) Clarifier with automatic sludge removers
- (c) Sludge Storage tank
- (d) Communitron
- (e) Stand-by Blower
- (f) Splitter box for recirculation of excess flow to pumping station No. 2
- (g) Electrical control panel containing starters, breakers, switches and all other controls needed to automatically control all plant equipment
- (h) An audible alarm system to be activated upon plant malfunction
- (i) Grating over whole plant area.

2.9.06 CORROSION PROTECTION

Internal and external painting of the tanks, pipes, fittings and valves shall be in accordance with the manufacturers standard practice. To prevent excessive galvanic corrosion cathodic protection shall be provided for the tanks.

2.9.07 INSTALLATION

The Contractor shall supply the Services of a Commissioning Engineer to check the condition of the equipment on arrival and after the installation is completed. The commissioning Engineer shall set the equipment in operation and instruct the Corporation personnel in the operation and maintenance of the equipment.

2.9.08 SPARE PARTS

The Contractor shall supply a list with prices of recommended spare parts which will enable maintenance and repair work to be carried out with minimum disruption to the continuous operation of the plant.



## ANNEX I

### EVAPORATION FROM COOKSON RESERVOIR

Annex I contains copies of the following two reports:

- (1) "Evaluation of Evaporative Water Loss Estimates for Cookson Reservoir on the East Poplar River", by James F. MacLaren Limited and J.E. Edinger Associates Inc., December 1978.
- (2) "Evaporation from Poplar River Cooling Reservoir", by Mostafa A. Shirazi, April 1977.

Both reports have been re-typed to reduce space.



EVALUATION OF EVAPORATIVE LOSS  
ESTIMATES FOR COOKSON RESERVOIR  
EAST POPLAR RIVER

for

ENVIRONMENT CANADA  
ENVIRONMENTAL PROTECTION SERVICE

DECEMBER 1978

JAMES F. MacLAREN LIMITED  
J. E. EDINGER ASSOCIATES INC.

**JAMES F. MacLAREN LIMITED**

CONSULTING ENGINEERS, PLANNERS AND SCIENTISTS

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11243-0

19 December 1978

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Head, Thermal Pollution and  
Special Studies Section  
Water Pollution Programs Branch  
Water Pollution Control Directorate  
Environmental Protection Services  
Ottawa, Ontario  
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Dear Mr. Draper:

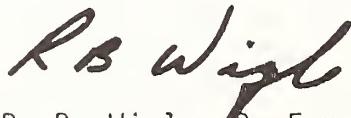
Attached is our review of "Evaluation of Evaporative Water Loss Estimates for Cookson Reservoir on the East Poplar River" prepared in conjunction with J. E. Edinger Associates Inc.

In carrying out the investigations, personal communications were made with Professor S. Solomon of the University of Waterloo, Mr. D. Davis of the Inland Waters Directorate, Department of the Environment, Mr. B. Godwin from the Prairie Farm Rehabilitation Administration, Mr. D. Turgeon of the Water Survey of Canada Branch, Department of the Environment, Dr. L. D. Spraggs from McGill University and Dr. M. A. Shirazi from E.P.A. Their comments were very helpful in assessing the various aspects of the study.

I take this opportunity to thank you for the privilege of carrying out this investigation and your assistance in the definition of the work is sincerely appreciated.

Yours sincerely,

JAMES F. MacLAREN LIMITED



R. B. Wigle, P. Eng.  
Manager, Hydrotechnical Division



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Encl.

## EVALUATION OF EVAPORATIVE WATER LOSS ESTIMATES FOR COOKSON RESERVOIR ON THE EAST POPLAR RIVER

### Introduction

A review of evaporative loss estimates given in various reports for Cookson Reservoir on the East Poplar River was requested by the Plant, Mine and Reservoir Operations Committee of the International Poplar River Water Quality Board. The Board was established by the International Joint Commission (IJC) to examine possible transboundary water pollution problems with the Poplar River thermal power plant and cooling reservoir. Estimates of evaporative water loss are to be used in a reservoir simulation model to compute reservoir water quality and availability of water for release across the International Boundary. Specifically, it has been requested that the bases of the various estimates be identified and differences be evaluated.

The Surface Water Quality Committee, another committee under the IJC Board, is attempting to simulate water quality conditions for the years 1931 to 1974 as they would have existed with present projects and flow apportionments in operation. The results of the water quality simulations are to be used to assess the need for mitigation measures to maintain downstream water quality objectives.

The Plant, Mine and Reservoir Operations Committee has two main studies on evaporative water loss from Cookson Reservoir. These are:

- (1) Poplar River Reservoir Final Environmental Assessment for Saskatchewan Power Corporation prepared by Saskmont Engineering (1978); and
- (2) Evaporation from Poplar River Cooling Reservoir prepared for the U.S. Environmental Protection Agency (EPA) by M.A. Shirazi (1977). Additional work in this field done by Sheppard T. Powell Associates was not finalized in a report (Scott, p.c.). The Saskmont Engineering report incorporates the evaporative loss estimates derived by Spraggs (1978) in Appendix E and the hydrology or water budget model used by Montreal Engineering Ltd. in Appendix B.

### Comparison of Saskmont and EPA Results and Methods

The Saskmont Engineering studies simulate evaporative losses on a monthly basis for the 1931 to 1974 period while the EPA study simulates evaporative losses for a one-year period. Thus, the results of the Saskmont studies are compatible with the period of studies to be used in the simulations being conducted by the IJC committees.

The Saskmont studies are based on reservoir temperature simulations using a one-dimensional, planar homogeneous time-varying reservoir model for natural temperatures and natural evaporation which is augmented by separate forced evaporation estimates for steam plant operations. The EPA estimates are based on a year of observed water temperatures from another project which are augmented by an estimated excess temperature for steam plant operations. The Saskmont estimates of natural evaporation are computed from water surface temperatures for a fixed reservoir water surface area and forced evaporation is computed from a surface heat budget which gives evaporative heat loss proportional to plant heat rejection. The EPA estimates are made for two different reservoir surface areas of 1200 acres and 700 acres. Saskmont also performs sensitivity analyses of excess temperatures and total evaporation as a function of reservoir elevations.

There appears to be discrepancies between the overall comparative estimates prepared by Saskmont and EPA which can be identified by comparing the Spraggs (1978) Table IV-10 yearly total results with the Shirazi (1976) summary statements as follows:

	<u>Saskmont: 1620 Acres</u>	<u>EPA: 700 Acres</u>	<u>EPA: 1200 Acres</u>
Natural (ac-ft)	5,039	1,960	3,360
Forced (ac-ft)	2,069	-	-
Total (ac-ft)	7,108	7,730 <sup>1</sup>	9,120 <sup>1</sup>
Total (ac-ft)	-	8,500 <sup>2</sup>	11,300 <sup>2</sup>

The totals given for Saskmont are for the Brady evaporation wind speed function and for EPA, a different wind speed function. Since the EPA natural evaporation estimates are proportional to surface area, they would give 4536 acre-feet of natural evaporation at 1620 acres which is 10% lower than the Saskmont estimates. The EPA estimates, however, give 5770 acre-feet of forced evaporation for 700 acres of surface area and 5760 acre-feet for 1200 acres (total minus natural) while the Saskmont estimate of forced evaporation is 2602 acre-feet. The EPA estimates show that forced evaporation is somewhat independent of surface area. Thus, there is an obvious discrepancy between the estimates of forced evaporation due to steam plant operations.

The discrepancies in the forced evaporation computations may be due to:

- 1) Use of different meteorological data, both year and stations;
- 2) Different natural lake temperatures
- 3) Different steam plant heat rejection rates
- 4) Different estimates of excess temperatures due to steam plant operations.

Since the natural evaporation rates are similar when converted to a common surface area basis, the differences due to the meteorological data are small. For the same reason, the natural lake temperatures used in the natural evaporation estimates must have been similar.

For heat rejection, Spraggs (1978) in Table IV-10 uses 87% for two-unit, 300 MWe plant operation with one-unit operation in June and July while Shirazi (1976) uses a constant heat rejection of  $5.89 \times 10^9$  Btu/hr for four 300 MWe units (4908 Btu/kw-hr). Since forced evaporation is approximately proportional to heat rejection and since Shirazi (1976) uses four-units versus two-units used by Spraggs (1978), it would be expected that the Shirazi estimates of forced evaporation are high by at least a factor of two. For two-unit operation, the Shirazi forced evaporation would be 2885 acre-feet. Furthermore, assuming 87% operation over ten months and 43% operation over two months reduces the Shirazi estimate to 2298 acre-feet which is less than the Spraggs estimate. Thus, when corrected to a common basis of heat rejection, the two yearly total estimates of forced evaporation are similar.

Excess temperatures are estimated by Shirazi (1976) using a plug flow temperature decay and averaging the plant inlet and discharge temperatures. Spraggs (1978) provides two estimates: one based on a completely mixed pond formula added to the natural temperatures computed from the planar homogeneous reservoir model; and another based on two-dimensional vertically mixed hydrodynamic computations. The Spraggs (1978) results for the two methods given in his Table IV-12 show a difference in forced evaporation results of 793 acre-feet which is 10% of the total evaporation but is 38% of the forced evaporation. Thus, the Spraggs (1978) comparison shows that the forced evaporation estimates are sensitive to the excess temperature estimates.

The EPA and Saskmont total evaporation estimates are similar when corrected for surface area and heat rejection rate. As shown above, however, the total evaporation is most sensitive to the reservoir operating surface area. Increased surface area increases the area available for natural evaporation and may increase natural water surface temperatures. Increased surface area decreases excess temperatures for a given heat rejection rate and is the primary reason why forced evaporation is insensitive to surface area. Thus, two factors that need further evaluation are: (1) the proper reservoir surface areas to use in the natural evaporation estimates; and, (2) the proper excess temperatures to use in the forced evaporation computations.

#### Surface Area and Evaporation

The Spraggs (1978) natural evaporation estimates based on constant surface area were converted to unit surface area natural evaporation rates as given in his Table IV-7 and were used in the Montreal Engineering water budget and reservoir level computations after addition of the constant monthly forced evaporation given in Table IV-10. Thus, the evaporative loss computations were performed first for a constant surface area and then used to compute the variation in surface area. Since natural temperatures and excess temperatures change with surface area, the two computations of the water budget and heat budget should be performed simultaneously rather than independently.

Since completion of the Saskmont report, Spraggs (1978 p.c.) has performed natural evaporation computations for different constant lake levels throughout the year and finds that natural lake surface temperatures are increased by about 1 deg. C. He reports, however, that there is no difference in monthly rates per unit surface area for the two different surface areas. This coincides with the Shirazi (1977) mean annual natural evaporation results for different lake areas shown earlier.

The evaporation wind speed function used by Spraggs (1978 p.c.) in the one-dimensional lake model (Table IV-7) and the monthly forced evaporation computations (Table IV-10) was derived by him from local evaporation pan data. It is similar to the mean weekly Lake Hefner formula and gives zero evaporation at zero wind speed. Spraggs (1978 p.c.) found that the Brady wind speed function overestimated evaporation in comparison to evaporation pan data. It should be noted from the Shirazi (1977) EPA estimates given above that different windspeed functions can change the natural evaporation estimates by about 100%. It is evident, therefore, that the most sensitive assumption in the computations is the choice of evaporative wind speed functions.

For forced evaporation computations by month in Table IV-10, Spraggs (1978 p.c.) reports that the meteorological parameters (wind speed and dew point) and reservoir surface temperatures, and consequently the vapour pressure-water temperature slope (b), were varied from month to month for 1975 data. His computations show, as expected, that forced evaporation is insensitive to surface area. The 1975 results in Table IV-10 were applied over all years in the Montreal Engineering water budget simulations. As long as the monthly plant load remains the same from year to year, then the only year to year variation in forced evaporation for each month would be due to meteorological data. Spraggs (1978 p.c.) estimates that its variation from year to year would be 10 to 15%. It should be no larger than the percentage difference in natural evaporation computed for each month for the two different meteorological stations in Table IV-10.

The one dimensional natural temperature reservoir model has the capability of representing stratification. Spraggs (1978 p.c.) has included a wind speed dependent vertical dispersion calibrated against Boundary Reservoir data. It results in no stratification for the wind speeds at Cookson Reservoir. Detailed longitudinal-vertical hydrodynamic computations with the plant in operation were not successfully completed by Spraggs (1978 p.c.). However, direct measurements at Cookson Reservoir when the first steam unit is in service should help to define and clarify the stratification question.

#### Comprehensive Water Budget

There is some question of how precipitation, evapotranspiration, and evaporation from Cookson Reservoir should be used to modify the observed flows downstream of the project in obtaining estimates of flows during reservoir and steam plant operation. This concern can be addressed by evaluating the water budget components before and after project operation.

The observed flows at East Poplar River Station 11AE003 over the period 1931 to 1974 are composed of inflows at the reservoir boundaries and runoff within the area inundated by the reservoir:

$$Q_o = Q_{in} + R \quad (1)$$

where  $Q_o$  is the flow observed at station 11AE003;

$Q_{in}$  is East Poplar River flow at the reservoir upper boundary plus any tributary inflow beyond the reservoir boundary;

$R$  is runoff contributed by the area inundated by the reservoir.

Values of  $Q_o$  for winter conditions taken from the report "Joint Studies for Flow Apportionment, Poplar River Basin in Saskatchewan and Montana" may be in error for the following reasons:

- 1) Winter flows were tabulated at Station 11AE003 for the period 1936-7. However, the records have not had their background completely researched (Godwin, Turgeon, p.c.) and as such, their accuracy has not been established.
- 2) Winter flows at station 11AE003 have been synthesized using recorded winter flows at a station further downstream (IJC Joint Studies for Flow Apportionment, 1976) with a drainage area approximately 10 times greater than that of station

11AE003. While this may be appropriate for determining flow apportionment, the use of these winter flows in operational studies should be made with caution. This extrapolation of winter flows from a downstream region to an upstream region for which the groundwater conditions may differ greatly requires field studies to determine its validity (Solomon, p.c.).

- 3) During the winter months, flows have been observed to be present but only in very small quantities due to groundwater sources (Godwin, p.c.). Others have observed the river to freeze completely during the winter months thereby eliminating flow (Turgeon, p.c.). The assumption of zero winter flow was made in other studies done on prairie watersheds (Solomon, p.c.).

The values of  $Q_0$  presented in the report "Joint Studies for Flow Apportionment, Poplar River Basin Saskatchewan and Montana" represent a compromise between USGS and Canadian authorities for flow apportionment and as such, their literal use in operational studies should be made with caution.

Runoff from the area inundated by the reservoir may be evaluated as:

$$R = (P - E_{VT} - I)A \quad (2)$$

Where  $P$  is precipitation

$E_{VT}$  is evapotranspiration

$I$  is infiltration, and

$A$  is reservoir area.

Post-operational flows can be written

$$Q'_o = Q_{in} + (P(1-f) - E_T - S_p) A + \Delta S \quad (3)$$

where  $Q'_o$  is post-operational flow;

$f$  is a fraction ( $f \leq 1$ ) representing immediate evaporation of precipitation over the heated water surface;

$E_T$  is total evaporation;

$S_p$  is groundwater seepage from the reservoir; and

$\Delta S$  is change in reservoir volume and may be either negative (storage) or positive (releases) and will be determined from the Poplar River Reservoir operating schedule.

Subtracting equation 1 from equation 3 and solving for  $Q'_o$  yields,

$$Q'_o = Q_o - R + (P(1-f) - E_T - S_p) A + \Delta S \quad (4)$$

Spraggs (1978) shows in Table IV-7 that total mean precipitation is 0.37 in., or 1970 ac-ft for the entire reservoir. In pre-impoundment conditions (Eq.2), this amount is reduced by evapotranspiration and infiltration. In order to properly include precipitation in the water budget estimates it would be necessary to estimate this factor.

Evapotranspiration can be estimated from the Turc formula, which has the advantage of empirically relating evapotranspiration to available precipitation (Solomon p.c.). This would give an estimate of  $c$  in the following relation.

$$R = cPA \quad (5)$$

Equation (4) can be rewritten

$$Q'_o = Q_o + P(1 - f - c)A - E_T A - S_p A + \Delta S \quad (6)$$

By assuming  $f = 0$  and  $c$  for the reservoir equal to the basinwide  $c$  of less than 10 percent, the procedure in the Saskmont report does not provide a conservative estimate of evaporation.

The net effect of precipitation on the reservoir surface is also much less than the evaporation term. It is therefore possible that the magnitude of the error in  $E_T$  estimates due to uncertainties in the windspeed function could be a significant percentage of the net effect on precipitation.

In view of the errors associated with the terms  $Q_o$  and  $E_T$ , however, and the lack of data for the groundwater seepage term,  $S_p$ , a refined calculation of the effects of precipitation are not warranted at this time.  $S_p$  is the only remaining unknown of possible significance in the water budget.

### Conclusions

1. The evaporation estimates presented as a time-series by Spraggs (1978) are adequate for the IJC water quality modelling studies. The effect on the overall water budget of computing natural evaporation independently of changes in reservoir surface area is negligible compared to the uncertainties introduced by the choice of the windspeed function. Spraggs' method of evaluating this function, however, is more realistic than any of the other methods, as it is based on nearby pan evaporation data.
2. The discrepancies between the Shirazi estimates and those made by Spraggs have been found to be based on the following:
  - a) Different assumed surface areas - when put on unit area basis, the Shirazi and Spraggs estimates of natural evaporation differ only by 10%.
  - b) Different assumed heat loads - when reported on the basis of equal heat rejection, the Spraggs and Shirazi estimates of forced evaporation again differ by only 10%. This discrepancy may be due to differences in estimating excess temperature.
3. More significant sources of error than those found in the evaporation estimates are the neglect of the induced groundwater seepage and uncertainties in reported winter flows. The method of including precipitation in Spraggs' estimates results in a somewhat non-conservative estimate of the effect on the water budget.

4. It is recommended that the water quality simulations be started using the existing evaporative losses in the Saskmont Report. More detailed studies concerning reservoir water budget would be required only if sensitivity analyses conducted with the water quality model indicate that changes in winter flow regime and seepage are significant.
5. A more comprehensive water budget and water quality study would require the following data:
  - a) Spring, summer, and fall vertical temperature profiles of Cookson Reservoir should be obtained to validate the one-dimensional, planar homogeneous model applied to the reservoir.
  - b) Validation of the flows for the years 1931-1974 reported at station 11AE003 can be made by observation of current flows relative to the downstream gauges. Estimates of reservoir seepage can also be made at that time.
6. Should a more detailed water budget be required, the time series of flow and meteorological data should be used to evaluate all the terms on the right-hand side of Equation 6. This effort may involve a modification of the one-dimensional model used to compute natural evaporation and release temperatures to be in agreement with field observations.

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EVAPORATION FROM POPLAR  
RIVER COOLING RESERVOIR

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Prepared for EPA Region VIII  
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## I. Introduction

An electric generation project proposed by the Saskatchewan Power Corporation impounds the East Branch of Poplar River to form a cooling reservoir. Since this is an international river and the proposed dam would be located within five miles of the United States-Canada border, the project raises water quality concerns to the State of Montana and to Environmental Protection Agency Region VIII.

The impoundment of the river and the use of the reservoir for cooling within the Canadian border is subject to certain water apportionment agreements as well as maintaining acceptable water quality. Evaporative water losses provide a first order link between water quantity and quality. It is important, therefore, to establish an estimate of such loss at the outset.

Adequate details are outlined in this report to enable assessment of assumptions and comparison with other methods.

## II. Meteorological Data

Saskatchewan Power Authority supplies (a) dew point temperature, (b) solar radiation, (c) natural water surface temperature and (d) wind speed for Coronach, Province of Saskatchewan, corresponding to the geographical location of the proposed reservoir within the Canadian border. These data are listed in Table 1 and will be used for all calculations in this report.

## III. Engineering Data

There are four coal-operated units, each generating 300 megawatts of electric power. They totally reject 5.89 billion Btu/hr at  $11.67^{\circ}\text{C}$  ( $21^{\circ}\text{F}$ ) condenser  $\Delta T$ . The volume flow rate of cooling water is  $34.4 \text{ m}^3/\text{s}$ . The reservoir when flooded to its highest elevation is nearly 1800 acres (728 ha). Calculations are performed for a reservoir area as small as 700 acres (283 ha). The distance between intake and discharge, measured along the river bed, is about 5 km.

The discharge is a rectangular, open end conduit  $3 \times 5$  meters with an invert elevation of 739 m. There is no skimmer wall considered at the intake end of the reservoir.

## IV. Intermediate Calculations

Table 2 shows a list of items that are common to all forthcoming calculations. These items are explained below:

- (a) Wind speed function  $f(W)$  - Heat loss by evaporation and by conduction from the water surface are strong functions of ambient air flow. The greater the wind speed, the greater

would be both losses. The functional form of wind speed for evaporation losses is not well known. Numerous wind functions have been compared by Edinger and Brady (1974). For example,

$$f_1(W) = 9.2 + 0.46 W^2 \text{ Wm}^{-2} \text{ mmHg}^{-1}$$

where  $W$  is wind speed in  $\text{ms}^{-1}$  and  $f_1(W)$  is expressed in Watts ( $\text{W}$ ) per  $\text{m}^2$  per  $\text{mmHg}$ .

For the range of wind speed used in this work, the above wind function yields low values. Higher, but realistic values of wind speed function were obtained from visual inspection of upper bound of data presented in Reference 2 and identified by  $f_2(W)$  in Table 2. Both quantities will be used in our calculations, yielding conservative and more liberal evaporation estimates for the purpose of comparison.

- (b) Humidity ratio w - This is defined as the ratio of mass of vapor per unit mass of dry air. It was obtained from a standard psychrometric chart, from the intersection of the dew point temperature and saturated vapor line.
- (c) Ambient air vapor pressure  $e_a$  - This was obtained from the definition of humidity ratio for a mixture of water vapor at  $e_a$  and air water vapor mixture at a standard pressure  $e = 760 \text{ mmHg}$ , i.e.

$$w = \frac{R_a e_a}{R_s 760}$$

$$e_a = \frac{(85.7)760}{53.3} = 1222w \text{ mmHg}$$

where  $R_a$  and  $R_s$  are, respectively, the universal gas constants for air and for steam.

## V. Natural Evaporation

Heat loss by evaporation per unit area can be calculated from:

$$H_e = \beta(T_s - T_d) f(W), \text{ Wm}^{-2}$$

$$\beta = .35 + .015 T_m + .0012 T_m^2, (\text{mmHg}/^\circ\text{C})$$

$$T_m = (T_s + T_d)/2, {}^\circ\text{C}$$

$$T_s = \text{Surface water temperature, } {}^\circ\text{C}$$

$$T_d = \text{Dew point temperature, } {}^\circ\text{C}$$

For total natural evaporation,  $H_e$  is multiplied by the surface area either of the reservoir or the area of the 5 km stretch of the river without impoundment. The heat loss  $H_e$  can also be calculated more directly from

$$H_e = (e_s - e_a) f(W), \text{ Wm}^{-2}$$

where  $e_s$  is the saturated vapor pressure at the surface water temperature;  $e_s$  is found from the standard steam tables.

A graph of this function is shown plotted in Figure 1. Also on the same figure, one finds a graph of the slope of  $e_s$  with temperature. The latter will be needed in the calculations of forced evaporation. A listing of heat lost from water surfaces under natural conditions can be found in Table 3.

In order to calculate the equivalent rate of water volume escaping from the surface in the form of vapor, we need to multiply  $H_e$  by the conversion factors .0408 and 0.0699, respectively, for the 700 acre and 1200 acre ponds. The results are thus expressed in cubic feet per second which are pro-rated for the entire year even though water loss by evaporation occurs only from April through October. Under natural conditions, surface ice is present from November through March.

## VI. Forced Evaporation

Induced heating of the water results in excess evaporation above the natural evaporation by a magnitude that is proportional to the change in the vapor pressure. An estimate of the vapor pressure above a heated reservoir can be obtained from the knowledge of the effective surface temperature  $T_s$  of the reservoir. We shall assume for the purpose of these calculations that  $T_s$  is an average temperature between the intake temperature  $T_i$  and the discharge temperature  $(\Delta T + T_i)$ , i.e.

$$T_s = \frac{T_i (\Delta T + T_i)}{2}$$

where  $\Delta T$  is the temperature differential across the condenser taken to be  $11.67^{\circ}\text{C}$ .

The intake temperature is an important parameter. It must be calculated from meteorological and engineering data. Ideally, details of temperature distribution from intake to discharge must be predicted to account for the reservoir geometry, discharge condition, and induced flow circulation. Since this is very difficult to do, simplifying assumptions that are typical for these problems are made to find an estimate of the temperature. Accordingly, it is assumed that the temperature decays exponentially from discharge to intake, i.e., there exist no vertical and lateral temperature gradients in any cross section along the reservoir. Temperature gradient exists only along the length of the reservoir, decreasing from discharge to the intake.

This longitudinal temperature decay depends on the equilibrium temperature  $T_e$ , the flow rate  $Q$ , the surface area  $A$ , the heat exchange coefficient  $K$ , water density  $\rho$  and its specific heat  $C_p$ . Thus,

$$\frac{T_i - T_e}{T_o - T_e} = e^{-r}$$

where  $r = KA/\rho C_p Q$  and  $T_o = T_i + \Delta T$

Using approximate expressions for  $K$  and  $T_e$  given in Reference 2, we have

$$K = 4.5 + .05T_s + f(W) \left( \frac{de_s}{dT} + .47 \right)$$

and

$$T_e = T_d + H_s/K$$

where  $T_d$  is the dew point temperature and  $H_s$  the solar radiation.

Calculations are straightforward except for the fact that  $K$  is dependent on  $T_s$  which cannot be calculated without the unknown  $T_i$ . For this reason initial estimates must be made of  $T_s$  and inserted for the equations for  $K$ . Corresponding initial estimates are then obtained for the equilibrium temperature  $T_e$  and the intake temperature  $T_i$ . The latter will be used to obtain better values of  $T_s$  and  $K$ . In this manner, iterations continue until convergence is obtained. Results of such calculations for 700 and 1200 acre ponds are presented respectively in Tables 4 and 5.

Lower values of wind speed function [i.e.,  $f_1(W)$ ] were used in these tables. To compare similar results obtained with the higher value of wind speed function,  $f_2(w)$ , Tables 6 and 7 were prepared.

The Summary of all results is presented in the next section.

## VII. Summary and Discussion of Results

1. The average natural evaporation during seven months, April through October, is 8.1 cfs for a 1200 acre impoundment and 4.7 cfs for a 700 acre impoundment. These translate to 3360 and 1960 acre-feet per year respectively, for 1200 and 700 acre impoundments. If there were no impoundments at all, the average loss for the same period is 0.6 cfs or 250 acre-foot per year. The yearly average is less than the seven month average by a factor of 1.7. See Table 3.

2. The average yearly total evaporation for a 1200 acre impoundment is at least 12.8 cfs or 9120 acre-foot. See Table 5.

3. The average yearly total evaporation for a 700 acre impoundment is at least 10.8 cfs or 7730 acre-foot. See Table 4.

4. More conservative estimates of the total evaporation using appropriate wind functions [i.e.  $f_2(w)$ ] are: 11,300 acre-foot (15.8 cfs) and 8500 acre-foot (11.9 cfs) respectively, for 1200 acre and 700 acre reservoirs. These values were obtained indirectly from Tables 7 and 6. Since partial calculations were made in these tables, comparisons were made with corresponding months in Tables 5 and 6 and average yearly evaporation from these tables was increased proportionally.

5. The above do not include effects of recirculation. It is difficult to say, but recirculation of uncooled water is unlikely to become a problem with the given design. Recirculation, if it occurs, will increase evaporation, however.

6. The estimates presented in this report are lower than those presented in Reference 1, making the latter estimates adequately conservative. No calculations were performed for sites B and C at this time.

7. Reference 1 did not present documentation of the method used. In this report, such documentation is presented for the method used. Additional calculations can be made, of course, to check new conditions. Considerably more effort is required to achieve a relatively small improvement in the accuracy of the estimates.

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TABLE 1 Meteorological Data for Coronach, Site A, Data Obtained from Reference 1

	January	February	March	April	May	June	July	August	September	October	November
Dew point $^{\circ}\text{C}$ $(^{\circ}\text{F})$	-16.94 (1.5)	-13.61 (7.5)	-8.06 (17.5)	-2.50 (27.5)	2.78 (37)	8.33 (47)	11.39 (52.5)	9.72 (49.5)	4.72 (40.5)	0 (32)	-7.5 (18.5)
Solar radiation Langleys, Hs	128	213	350	425	500	550	600	490	355	227	125
Wind speed Mph m/s	14.8 3.31	14.3 3.2	14 3.13	14.3 3.2	14.4 3.22	13 2.91	12 2.68	12 2.68	14 3.13	14 3.13	14.6 3.27
Natural water temperature $^{\circ}\text{C}$ $(^{\circ}\text{F})$	-14.17 6.49	-9.72 14.50	-3.17 26.29	3.61 38.50	9.44 48.99	15.28 59.50	19.17 66.51	17.18 62.92	12.11 53.80	5.56 42.00	-3.06 26.49
H-16											

TABLE 2 Intermediate Calculations

$f_1(W) \text{ W}^{-2} \text{ mmHg}^{-1}$	14.24	13.91	13.71	15.50	13.97	13.10	12.05	12.05	13.71	13.71	14.12
$f_2(W) \text{ W}^{-2} \text{ mmHg}^{-1}$	27.50	27.00	26.00	27.00	27.50	25.00	28.00	28.00	26.00	26.00	27.00
w	.0008	.00115	.00185	.003	.0046	.0068	.0082	.0075	.0053	.0037	.002
$e_a \text{ mmHg}$	.98	1.41	2.26	3.67	5.62	8.25	10.02	9.17	6.42	4.75	2.44

Table 3a. Heat loss  $H_e^*$  from Water Surface  
Under Natural Conditions.

$e_s$ mmHg	6.0	9.10	13.30	17.00	15.20	10.90	6.40
$H_e_1$ $\text{Wm}^{-2}$	33.96	45.26	63.19	83.60	78.92	56.84	30.56
$H_e_2$ $\text{Wm}^{-2}$	62.91	95.70	126.30	196.40	168.80	116.50	42.90

Table 3b. Yearly Averaged Water Loss CFS.\*\*

No Reservoir	700 Acre		1200 Acre		1800 Acre		
$Qe_1$	$Qe_2$	$Qe_1$	$Qe_2$	$Qe_1$	$Qe_2$	$Qe_1$	$Qe_2$
.18	.35	1.42	2.75	2.44	4.72	3.65	7.08

\*Indices 1 and 2 refer to different methods of calculation  
as explained in this section.

\*\*Evaporation rates during seven months are higher than  
indicated for the yearly average by a factor of  $12 \div 7 = 1.71$ .

Note that  $35.31 \text{ CFS} = 1\text{m}^3\text{s}^{-1}$   
 $1 \text{ acre-foot/year} = 1.401 \times 10^{-3} \text{ CFS}$

Table 4. Calculations for Site A, 700 Acres with Low Values of Wind Speed Function.

	January	February	March	April	May	June	July	August	September	October	November
$W_{MS}^{-1}$	3.31	3.20	3.13	3.20	3.22	2.91	2.68	2.68	3.13	3.13	3.27
$f_1(w) \text{ Wm}^{-2} \text{ mmHg}$	14.24	13.91	13.71	15.50	13.97	13.10	12.05	12.05	13.71	13.71	14.12
$H_S \text{ Wm}^{-2}$	62.00	103.00	169.40	205.70	242.0	266.2	290.4	237.20	171.80	109.90	60.50
$T_d \text{ }^{\circ}\text{C}$	-16.94	-13.61	-8.06	-2.50	2.78	8.33	11.39	9.72	4.72	0	-7.50
$w$	.00008	.00115	.00185	.003	.0046	.0068	.0082	.0075	.0053	.0037	.002
$e_a \text{ mmHg}$	.98	1.41	2.26	3.67	5.62	8.25	10.02	9.17	6.42	4.75	2.44
$T_s \text{ }^{\circ}\text{C}$	13.01	15.37	20.33	22.63	27.22	31.45	34.43	34.35	27.32	23.69	17.92
$de_s/dT \text{ mmHg}^{\text{-}1}$	.76	.86	1.12	1.26	1.61	1.98	2.30	2.31	1.64	1.34	.985
$T_e \text{ }^{\circ}\text{C}$	-14.20	-9.28	-1.86	3.84	9.71	15.30	18.72	18.70	9.66	3.60	-5.17
$T_i \text{ }^{\circ}\text{C}$	6.49	10.23	14.49	16.86	21.48	25.70	28.58	28.52	21.49	17.75	12.30
$K \text{ Wm}^{-2} \text{ C}^{-1}$	22.67	23.77	27.32	32.45	34.92	38.17	39.60	39.72	34.79	30.50	25.94
$e_s \text{ mmHg}$	11.40	13.30	18.00	20.50	27.30	35.00	41.50	41.30	27.50	22.00	15.80
$He_1 \text{ Wm}^{-2}$	148.38	165.39	215.80	260.87	302.87	350.43	379.33	387.17	289.01	236.50	188.64
$Qe_1 \text{ CFS}$	6.05	6.75	8.80	10.64	12.36	14.30	15.48	15.80	11.79	9.65	7.70

Table 5. Calculations for Site A, 1200 Acre and Low Values of Wind Speed Function

	January	February	March	April	May	June	July	August	September	October	November
$W \text{ m s}^{-1}$	3.31	3.20	3.13	3.20	3.22	2.91	2.68	2.68	3.13	2.13	3.27
$f_1(W) \text{ W}_m^{-2} \text{ mmHg}$	14.24	13.91	13.71	15.50	13.97	13.10	12.05	12.05	13.71	13.71	14.12
$H_s \text{ W/m}^2$	62.00	103.00	169.40	205.70	242.0	266.20	290.40	237.20	171.80	109.90	60.50
$T_d \text{ }^\circ\text{C}$	-16.94	-13.61	-8.06	-2.50	2.78	8.33	11.39	9.72	4.72	0	-7.50
$w$	.0008	.00115	.00185	.003	.0046	.0068	.0082	.0075	.0053	.0037	.002
$e_a \text{ mmHg}$	.98	1.41	2.26	3.67	5.62	8.25	10.02	9.17	6.42	4.52	2.44
$T_{sc} \text{ }^\circ\text{C mmHg-c}$	5.70	9.45	14.93	18.00	23.08	27.71	30.75	28.65	22.96	18.53	11.97
$de_s/dT$	.49	.61	.835	.99	1.30	1.64	1.92	1.73	1.28	1.02	.705
$T_d \text{ }^\circ\text{C}$	-13.58	-8.46	-.74	4.84	10.75	16.27	19.73	17.03	10.46	4.27	-4.70
$T_i \text{ }^\circ\text{C}$	-.12	3.62	9.09	12.22	17.25	21.81	24.92	22.88	17.13	12.70	6.13
$K \text{ W m}^{-1} \text{ C}^{-1}$	18.47	20.00	23.14	28.03	30.38	33.53	34.84	32.44	29.92	25.71	21.62
$e_s \text{ mmHg}$	6.80	8.90	13.00	16.00	21.00	28.00	33.50	30.00	21.00	16.50	10.50
$H_e \text{ W m}^{-2}$	82.88	104.20	147.25	191.12	214.86	258.73	282.93	251.00	199.89	165.25	113.81
$Q_e \text{ CFS}$	5.74	7.28	10.29	13.36	15.06	18.08	19.78	17.55	13.97	11.48	7.96

Table 6. Calculations for Site A, 700 Acre at Highest Values  
of Wind Speed Function.

	January	February	March	April	June	August	September	October
$f(W)$ $\text{Wm}^{-2}\text{mmHg}$	27.50	27.00	26.00	27.00	25.00	28.00	26.00	26.00
$T_s$ $^{\circ}\text{C}$	4.91	8.25	13.00	16.86	25.23	24.95	21.39	17.30
$d\epsilon_s/dT$ $\text{mmHg}^{\circ}\text{C}^{-1}$	.47	.56	.76	.925	1.46	1.42	1.18	.96
$T_e$ $^{\circ}\text{C}$	-14.91	-10.46	-3.53	2.28	13.26	13.76	8.26	2.58
$T_i$ $^{\circ}\text{C}$	.83	2.41	7.17	11.02	19.39	19.11	15.55	11.46
$K$ $\text{Wm}^{-2}\text{C}^{-1}$	30.60	32.72	37.40	43.01	54.01	58.67	48.47	42.54
$e_s$ $\text{mmHg}$	6.55	8.15	11.50	14.70	24.50	24.00	19.30	15.30
$H_e$ $\text{Wm}^{-2}$	153.20	182.00	240.20	297.90	406.25	415.24	334.88	280.28
$Q_e$ CFS	6.25	7.42	9.80	12.15	16.97	16.94	13.66	11.43

Table 7. Calculations for Site A, 1200 Acre at High  
Values of Wind Speed Function.

	May	July	November
$f_2(w) \text{ Wm}^{-2} \text{mmHg}$	27.50	28.00	27.00
$T_s \text{ } ^\circ\text{C}$	17.41	24.22	6.37
$de_s/dT \text{ mmHgc}^{-1}$	.96	1.38	.51
$T_e \text{ } ^\circ\text{C}$	8.23	16.44	-5.58
$T_i \text{ } ^\circ\text{C}$	11.57	18.39	.54
$K \text{ Wm}^{-2} \text{C}^{-1}$	44.42	57.51	31.55
$e_s \text{ mmHg}$	15.30	23.00	7.16
$He_2 \text{ Wm}^{-2}$	266.20	363.44	127.44
$Qe_2 \text{ CFS}$	18.61	25.40	8.91

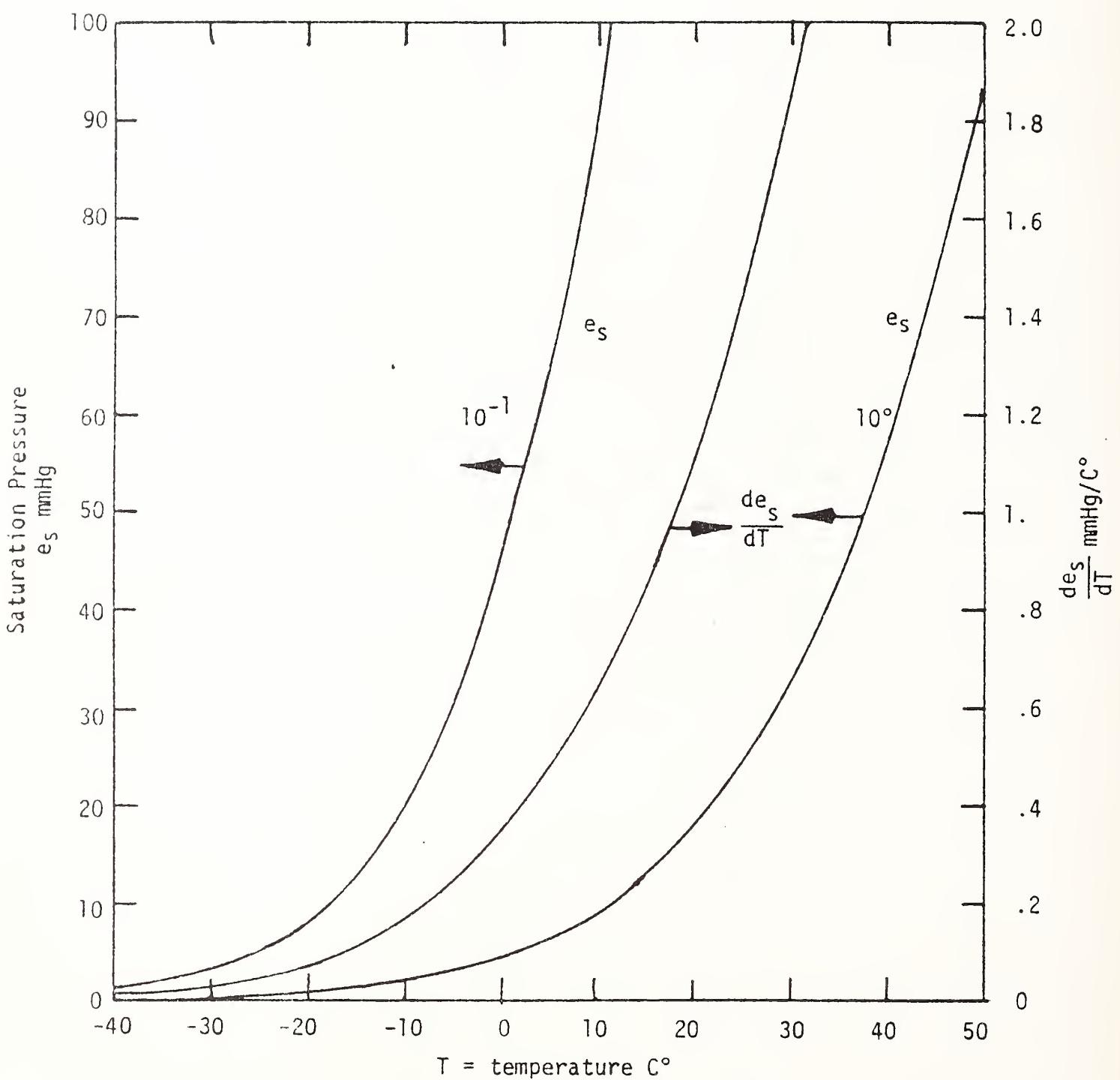


Figure 1: Saturated vapor pressure  $e_s$  and its gradient  $\frac{de_s}{dT}$  as a function of temperature

ANNEX J

RESERVOIR OPERATION MODEL DOCUMENTATION  
AND  
SELECTED MODEL OUTPUT LISTINGS

Annex J contains the following:

- (1) A computer model documentation prepared by Montreal Engineering Company Limited for the Plant, Mine and Reservoir Operations Committee. It has been re-typed to reduce space.
- (2) Copies of selected tables from Reservoir Operational Model Run Nos. 17B, 21B, 24 and 25.



## POPLAR RIVER RESERVOIR WATER QUALITY MODEL DOCUMENTATION

### Mainline Description

The conceptual flow chart for the mainline program of the Poplar River Reservoir Water Quality Model is given in Figure 1. The analysis is set up to be performed over a period of one month with the objective being to determine the water quality and quantity in the East Poplar River reservoir at monthend. When each year of study has been completed, twelve months of results are presented in tabular form. Provisions are made in the program for new input data to be read for each year's calculations.

The principal input information required to run the program is the following:

1. Initial volume and water quantity of reservoir.
2. East Poplar River monthly flows.
3. Monthly precipitation.
4. Monthly natural evaporation.
5. Monthly forced evaporation from reservoir.
6. Riparian releases.

Other data and constants used in the program are described where they occur in the mainline and subroutines.

There are nine principal chemical concentrations monitored by the program:

1. Total Alkalinity.
2. Calcium.
3. Magnesium.
4. Sodium-Potassium.
5. Sulphate.
6. Chloride.
7. Phosphate.
8. Total Nitrogen.
9. Boron.

From these principal parameters, the following variables are computed for the reservoir:

1. Total Dissolved Solids.
2. Sum of Ions.
3. Total Hardness.
4. pH Level.

The following mainline program description refers to the step numbers contained in the conceptual flow chart given in Figure 1.

1. The following program variables are initialized in data statements:
  - a) demineralization sulphate addition;
  - b) demineralization sodium addition;
  - c) Ryznar index for softening;
  - d) Langlier temperature correction;
  - e) pH temperature correction;
  - f) CO<sub>2</sub> concentration of reservoir;
  - g) reverse osmosis daily flow;
  - h) maximum volume of reservoir;
  - i) minimum volumes of reservoir;
  - j) summer chlorine addition;
  - k) winter chlorine addition;
  - l) station loss per day per unit;
  - m) mine water chemical composition.
2. The FACTOR subroutine is called to calculate the ratio of depth of runoff to depth of precipitation for each year.
3. The following input information is read from cards:
  - a) title of the study;
  - b) initial inflow year to be used;
  - c) the development year of the plant;
  - d) number of years in study;
  - e) initial reservoir volume;
  - f) water quality output file switch;
  - g) mine inflow switch;
  - h) softening process switch;
  - i) initial chemical concentrations of the reservoir.
4. The FLYASH subroutine is called to read a data card containing the initial volumes and states of the ash lagoons.
5. This is the beginning of the year loop which performs the water quality analysis for each year of the study.
6. At this point, provision has been made to redefine the number of generating units operating during the year. The default assumes the same units operating as in the previous year.
7. The RESFLW subroutine is called to read the monthly East Poplar River flows from the historic inflow disk file record corresponding to the current year of study.
8. If there is a mine water inflow then the AMINE function is used to determine the daily volume of mine water flow recognizing the stage of development of the plant.
9. The monthly arrays for the following variables are initialized to zero:

- a) softening acid addition;
  - b) softening lime addition;
  - c) reverse osmosis acid addition;
  - d) demand riparian release;
  - e) continuous riparian release;
  - f) reservoir spill;
  - g) ash lagoon outflow;
  - h) ash lagoon inflow.
10. This is the beginning of the month loop which calculates the reservoir water and chemical balances for each month of the study.
  11. The monthly mine water inflow volume is computed by multiplying the per diem rate by the number of days in the month.
  12. The volume of station losses for the month are calculated by multiplying the per diem rate by the days in the month and the number of units operating.
  13. The OUTFL subroutine is called to calculate the continuous and demand riparian releases from the reservoir during the month.
  14. The FLYASH subroutine is called to determine the volume of water diverted to and returning from the ash lagoons.
  15. A water balance is performed using the previously calculated inflows and diversions to compute a preliminary monthend reservoir volume.
  16. The EVAPO subroutine is called to iteratively calculate the monthly evaporation volume from the average surface area of the reservoir. The historic evaporation and precipitation disk file is used by EVAPO to calculate the net evaporation depth for the month.
  17. The final gross monthend reservoir volume is computed from the water balance equation using the inflow, diversions and evaporative losses calculated previously.
  18. If the gross monthend reservoir volume is less than the minimum allowed, then the number of units operating is reduced and the program branches to recalculate the reservoir losses.
  19. The INFCON subroutine is called to interpolate the values of the nine principal chemical parameters for the East Poplar River inflows, according to the concentration versus flow relationships stored in the CON array.
  20. The monthend chemical composition of the reservoir is computed by taking the weighted average of the reservoir water quality at the beginning of the month and the water quality of the inflows. Concentrations are given in mg/l and volumes are in  $m^3$  simplifying this calculation.

21. In this step the amount of sulphuric acid addition necessary for the reverse osmosis treatment of the boiler feed water makeup is computed. The quantity of acid addition is a complex function of the calcium, total alkalinity and total dissolved solids concentrations calculated previously. The amount of acid is varied until the mathematical function is satisfied, then the quantity of acid addition is stored for output.
22. The total alkalinity and sulphate concentrations of the reservoir are adjusted to reflect the sulphuric acid addition made in the reverse osmosis treatment.
23. Further adjustments are made to the reservoir's sulphate and sodium-potassium concentrations to take into account the spent demineralizer regenerates.
24. The chlorine addition for the condenser cooling water is also calculated according to the time of year to compute the resulting changes in total alkalinity and chloride concentrations in the reservoir.
25. If the RYZNAR index of the reservoir water is greater than 6, no softening is required.
26. This step of the program concerns the softening of the reservoir water by the addition of sulphuric acid and cold lime for scale control in the condenser cooling water. The quantity of the sulphuric acid and lime additions is a complex function of the RYZNAR index, the calcium, magnesium and TDS concentrations and the temperature of the return cooling water. The amount of softening acid is varied until the mathematical function is satisfied within certain limits. After this iterative process is finished, the resulting total lime addition is computed.
27. The reservoir phosphate, calcium, magnesium and total alkalinity concentrations are adjusted according to the softening additions. The total hardness, total dissolved solids and pH levels of the reservoir are also computed.
28. The preliminary net reservoir monthend volume including the riparian and demand releases is calculated.
29. If this monthend reservoir volume is greater than the maximum specified, then the resulting spill from the reservoir is determined.
30. The FLYASH subroutine is called to calculate the average water quality of the water taken from the reservoir to the ash lagoons during the month.
31. Unless the last month in the year has been reached the program loops back to begin the calculation of the next month's water quality.

4. OUTFL

The OUTFL subroutine determines the continuous and demand riparian releases from the Poplar River Reservoir.

5. FLYASH

The purpose of the FLYASH subroutine is to monitor the water quantity and quality in the flyash lagoons. The per diem diversion volume, the maximum volume of the ash lagoons, and the mg/l addition of boron are specified in data statements.

6. RESFLW

The RESFLW subroutine decodes the monthly East Poplar River inflows from the Historic Inflow File and calculates the total annual flow into the reservoir. The Historic Inflow File contains the International Joint Commission East Poplar River monthly flows in cfs for 1931 to 1976.

7. INFCON

The INFCON subroutine defines the chemical concentrations of the East Poplar River inflows as a function of inflow rate. The concentration versus inflow relationships for the nine principal chemical parameters are contained in a BLOCK DATA subprogram.

8. EVAPO

The EVAPO subroutine computes the total monthly evaporation volume from the reservoir as a function of the average surface area. The subroutine utilizes the Historic Evaporation and Precipitation File to determine the natural net evaporation for the month. The forced evaporation volume in each month for either one or two units operating is specified in a data statement.

9. ARVOL

The ARVOL function interpolates the surface area of the reservoir from the volume versus surface area relationship contained in data statements.

10. FACTOR

The FACTOR subroutine is used to reduce the precipitation that is assumed to have fallen on the watershed by the amount of precipitation which was recorded as runoff. This correction factor is applied to the precipitation that is considered to contribute directly to the reservoir volume as a function of the surface area.

32. The OUTPUT subroutine is called to print the yearly report containing all the monthly reservoir water balance, chemical concentration, acid and lime addition figures.
33. The reservoir water quality parameters are initialized and the inflow year and stage of development are incremented for the next year. If the number of years in the study has not been completed, then the mainline program branches to the beginning of the year loop, otherwise it stops.

#### Subroutines Description

##### 1. OUTPUT

This subroutine is used to prepare a yearly report containing the monthly values of the reservoir volume, inflows, releases, chemical concentrations and quantities of acid and lime addition. An example output is presented as Figure 2. The monthend reservoir volumes are given in  $m^3 \times 10^6$  while the monthly inflow and release volume are in  $m^3 \times 10^3$ . The monthend chemical concentrations are printed in mg/l and the quantities of acid and lime addition are given in metric tons.

If requested, the OUTPUT subroutine writes all the data contained in the yearly output report to a disk file for subsequent reprocessing of an entire run. Supplementary programs have been written to read and write this data in any format required including histograms and plotted output.

The OUTPUT subroutine also prints the Ash Lagoon Report shown as Figure 3. This report contains the total input volume and output volume for ash handling in  $m^3 \times 10^3$  with the years and months when the return flow is simulated to occur. In addition, the TDS concentrations of the return water and the reservoir are given in mg/l for the same months.

##### 2. AKTDS

The AKTDS function interpolates the Langelier TDS correction factor used in both the reverse osmosis and softening processes. The TDS versus Langelier correction relationship is contained in a data statement.

##### 3. AMINE

The AMINE function computes the pumped mine water inflow as a function of the plant development stage and the preceding annual East Poplar River inflow into the reservoir. The relationship between plant development and mine water inflow is given in a data statement.

POPLAR RIVER RESERVOIR WATER QUALITY MODEL FLOW CHART

FIGURE 1

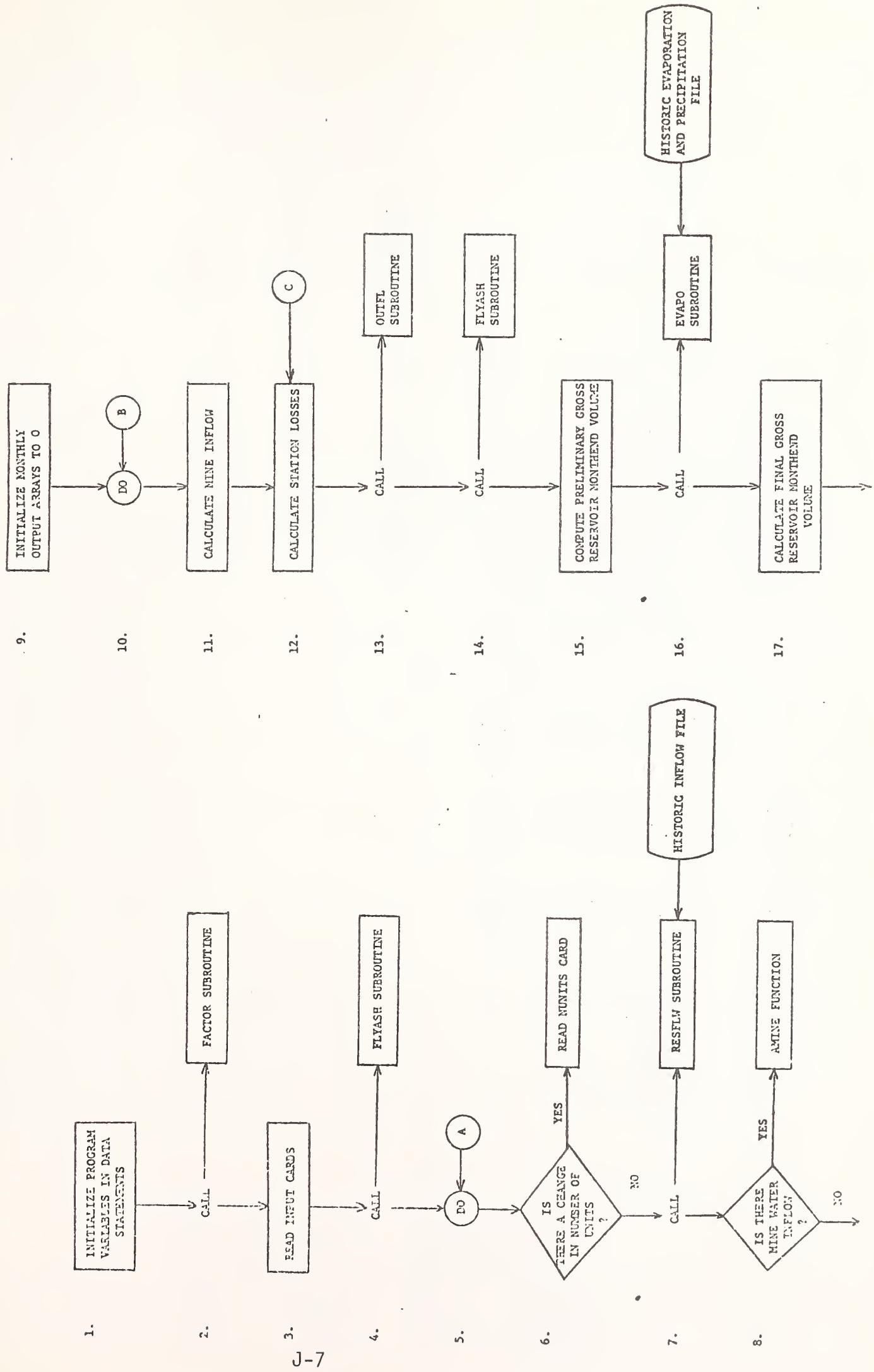


FIGURE 1 (continued)

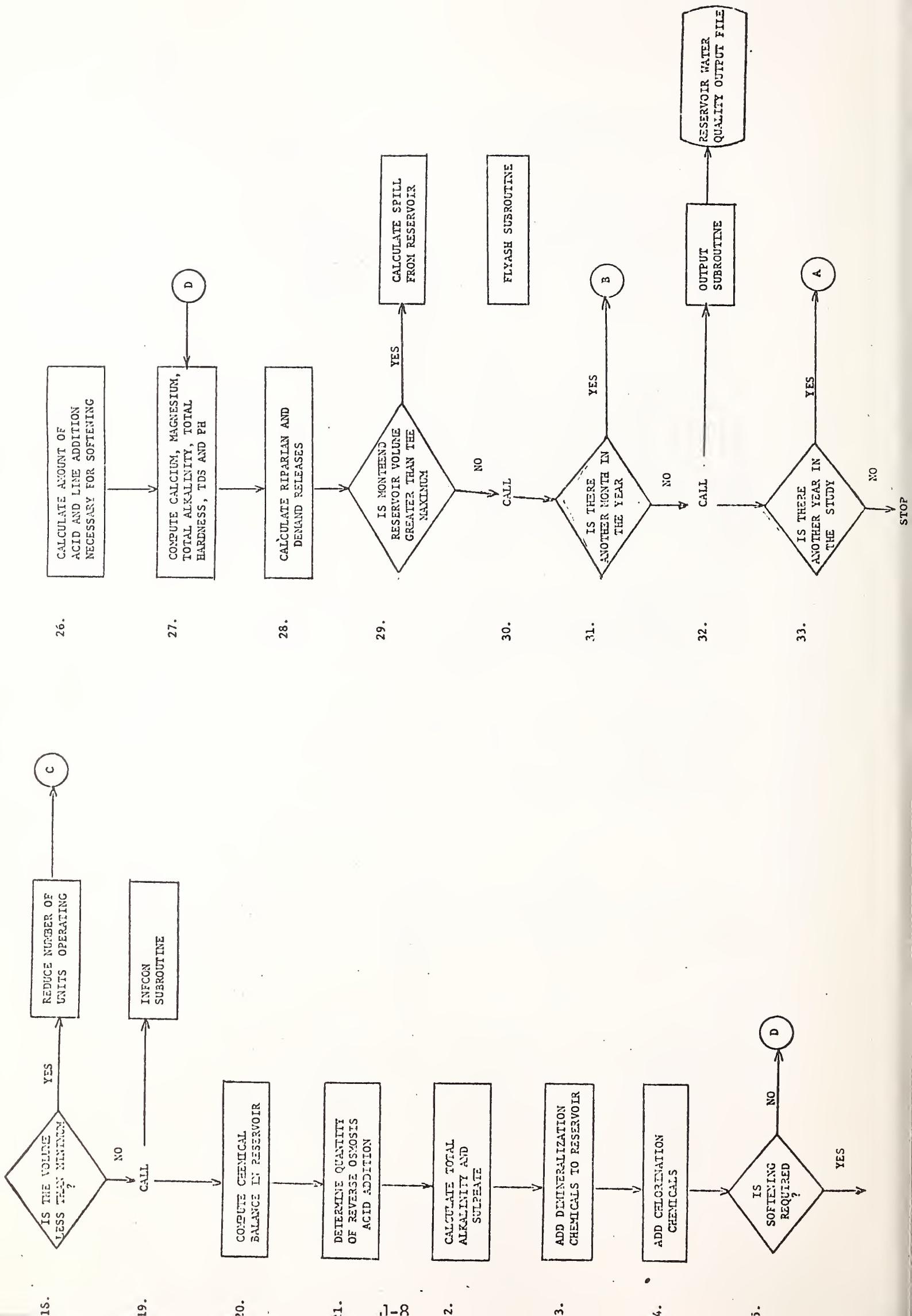


FIGURE 2

TWO UNIT OPERATION, GOOD WATER QUALITY START CONDITIONS, INC INFLOWS

## INFLOW YEAR 1959 - STAGE OF DEVELOPMENT 14

MONTH	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
TOTAL RIVER INFLOW	0.00	6.85	034.26	623.86	432.30	278.90	242.69	204.77	256.08	432.30	315.60	227.52
TOTAL RIVER INFLOW	86.67	78.28	86.67	83.87	86.67	83.87	86.67	83.87	86.67	83.87	83.87	86.67
TOTAL ASH WATER INFLOW	-427.37	1201.91	-427.37	-413.58	-427.37	1396.90	-427.37	-213.68	-413.58	567.76	-413.58	-427.37
STATION LOSS	10.14	9.16	10.14	9.81	10.14	4.91	10.14	5.07	9.81	10.14	9.81	10.14
EVAPORATION LOSS	98.42	115.40	121.79	20.30	1049.18	508.54	1548.08	1439.27	1070.18	600.58	71.41	96.87
STATUTORY RELEASE	-75.84	68.50	75.84	73.40	75.84	73.40	75.84	73.40	75.84	73.40	75.84	75.84
-CONTINUOUS	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
-ON DEMAND	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
OVERFLOW FROM RESERVOIR	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
RESERVOIR MONTH	-30.62	-31.71	-32.00	-32.18	-31.14	-32.31	-30.21	-28.76	-27.54	-27.77	-27.77	-27.47
END VOLUME	2	2	2	2	2	1	2	1	2	2	2	2
NUMBER OF UNITS OPERATING												
TOTAL HARDNESS	319.0	319.7	319.3	318.9	322.3	324.1	330.0	335.9	340.4	343.2	342.6	342.6
SUM OF IONS	1164.5	1169.1	1163.4	1158.4	1181.4	1193.0	1233.0	1274.6	1306.2	1324.3	1322.4	1322.9
TOTAL DISSOLVED SOLIDS	895.5	899.7	894.9	890.8	910.6	920.4	954.7	990.2	1017.6	1033.2	1033.2	1033.0
CALCIUM	97.0	96.9	97.4	97.9	96.2	95.5	92.9	90.2	88.3	87.3	87.4	87.7
MAGNESIUM	222.0	222.9	221.9	221.0	226.1	228.6	237.1	245.7	252.2	255.9	255.2	254.8
SODIUM-POTASSIUM	221.9	223.2	221.4	219.9	226.2	229.4	240.5	252.1	261.0	266.0	265.8	266.1
TOTAL ALKALINITY	433.8	434.4	433.0	431.7	436.8	439.6	448.9	458.6	465.5	469.4	468.2	471.7
SULPHATE	308.6	310.8	306.8	307.2	316.5	321.0	337.0	353.3	366.3	373.8	373.7	374.6
CHLORIDE	12.0	12.2	12.0	12.0	12.4	12.6	13.2	13.9	14.5	14.8	14.8	14.8
PHOSPHATE	0.210	0.209	0.205	0.202	0.200	0.198	0.190	0.194	0.197	0.194	0.194	0.194
NITRATE	3.304	3.321	3.274	3.232	3.305	3.336	3.479	3.627	3.735	3.747	3.764	3.765
BORON	1.47	1.47	1.46	1.45	1.44	1.51	1.59	1.61	1.73	1.77	1.78	1.78
REV. OSMOSIS. ACID	335.9	336.0	336.4	335.4	334.3	338.2	340.4	347.4	353.1	356.6	354.5	358.6
TOTAL RVO ACID	17.03	15.38	17.06	16.45	16.95	18.30	17.25	16.90	17.32	16.04	17.04	18.24
TOTAL LINE ADDED	36.06	59.24	53.19	38.18	263.70	154.12	367.02	350.61	280.83	239.99	86.97	79.28
SOFT. TOTAL ACID	7.08	11.72	10.38	7.46	49.84	29.76	68.86	66.20	53.68	46.44	17.10	15.74
ADDITIONAL ACID	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00

FIGURE 3

ASH LAGOON REPORT					
TWO UNIT OPERATION, GOOD WATER QUALITY START CONDITIONS, IJC INFLOWS					
YEAR	MONTH	INFLOW VOLUME	OUTFLOW VOLUME	OUTFLOW TDS	RESERVOIR TDS
1949	NOV	1054.64	716.05	1012.0	753.0
1950	MAR	847.84	821.15	775.4	757.8
1950	AUG	827.17	705.79	808.5	635.9
1951	JAN	1054.64	915.63	741.0	681.3
1951	MAY	834.06	789.52	719.1	660.8
1951	NOV	1054.64	854.57	834.5	712.4
1952	MAR	847.84	831.01	725.1	715.5
1952	AUG	827.17	648.25	763.5	489.6
1953	FEB	1902.48	1675.77	587.7	563.5
1953	JUN	1654.33	1645.31	569.5	615.4
1953	OCT	1275.21	1003.91	791.0	721.9
1954	FEB	1681.90	1594.03	753.2	730.5
1954	JUN	1654.33	1625.28	727.5	725.2
1954	OCT	1275.21	1111.38	822.7	820.3
1955	FEB	1681.90	1631.31	834.7	828.7
1955	JUN	1654.33	1602.31	795.3	512.8
1955	OCT	1275.21	998.59	660.6	608.1
1956	FEB	1681.90	1591.03	633.5	625.3
1956	JUN	1654.33	1578.85	649.5	653.4
1956	OCT	1275.21	1000.71	828.8	726.3
1957	FEB	1681.90	1601.33	756.4	737.3
1957	JUN	1654.33	1571.58	773.2	764.5
1957	OCT	1275.21	1020.17	956.3	835.4
1958	FEB	1681.90	1630.75	857.6	841.3
1958	JUN	1654.33	1539.40	868.6	800.0
1958	OCT	1275.21	937.42	1080.5	891.4
1959	FEB	1681.90	1587.92	933.0	899.7
1959	JUN	1654.33	1603.69	923.7	920.4
1959	OCT	1275.21	995.15	1188.5	1033.2
1960	FEB	1681.90	1615.71	1065.7	1037.6
1960	JUN	1654.33	1575.40	960.8	769.2
1960	OCT	1275.21	934.62	1046.5	896.7
1961	FEB	1681.90	1560.39	951.8	901.1
1961	JUN	1654.33	1544.79	953.3	919.2
1961	OCT	1275.21	865.15	1356.7	1060.8
1962	FEB	1681.90	1582.21	1113.3	1065.4
1962	JUN	1654.33	1564.97	1054.3	924.0
1962	OCT	1275.21	975.68	1201.3	1019.6

SELECTED TABLES FROM RESERVOIR OPERATIONAL MODEL  
RUN NUMBERS 17B, 21B, 24 and 25

## 2 UNITS. 1948(6000) START. NO SOFTENING, AVERAGE LEACHAGE, RUN 178.

NAME	YEAR	ICYR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
PES. ELEV.	1953	8	752.01	752.02	752.19	752.34	752.57	752.9	752.21	752.13	752.16			
PES. ELEV.	1954	9	752.16	752.20	752.54	752.87	752.97	752.50	752.45	752.38	752.40			
PES. ELEV.	1955	10	752.40	752.40	752.87	752.87	752.78	752.48	752.22	752.04	751.94	751.96	751.97	
PES. ELEV.	1956	11	751.97	751.97	752.33	752.61	752.52	752.45	751.16	751.97	751.80	751.73	751.75	
PES. ELEV.	1957	12	751.74	751.72	751.82	751.92	751.75	751.66	751.40	751.24	751.03	751.06	751.11	751.14
PES. ELEV.	1958	13	751.13	751.17	752.14	752.41	752.21	752.05	751.76	751.56	751.38	751.25	751.27	751.26
PES. ELEV.	1959	14	751.23	751.19	751.29	751.34	751.18	751.14	750.80	750.55	750.36	750.30	750.32	750.33
PES. ELEV.	1960	15	750.32	750.30	752.87	752.87	752.70	752.58	752.16	751.99	751.75	751.61	751.62	751.62
PES. ELEV.	1961	16	751.62	751.64	751.97	751.97	752.01	751.89	751.70	751.38	751.06	750.82	750.70	750.72
PES. ELEV.	1962	17	750.71	750.69	752.04	752.30	752.17	752.15	751.94	751.74	751.54	751.46	751.45	
PES. ELEV.	1963	18	751.45	751.64	752.53	752.66	752.54	752.62	752.19	751.98	751.79	751.61	751.60	751.60
PES. ELEV.	1964	19	751.60	751.60	751.62	752.08	751.91	751.80	751.45	751.20	751.07	750.91	750.91	750.92
PES. ELEV.	1965	20	750.91	750.89	750.89	751.96	751.97	751.99	751.72	751.58	751.51	751.40	751.43	751.43
PES. ELEV.	1966	21	751.42	751.41	752.04	752.08	751.91	751.89	751.55	751.38	751.16	751.08	751.11	
PES. ELEV.	1967	22	751.10	751.09	752.21	752.87	752.81	752.62	752.06	751.79	751.61	751.52	751.53	751.54
PES. ELEV.	1968	23	751.52	751.51	752.77	752.81	752.63	752.48	752.11	751.97	751.87	751.76	751.76	751.77
PES. ELEV.	1969	24	751.76	751.73	751.73	752.97	752.97	752.74	752.55	752.21	751.91	751.67	751.66	751.67
PES. ELEV.	1970	25	751.68	751.68	752.11	752.87	752.87	752.87	752.45	752.15	751.97	751.87	751.87	
PES. ELEV.	1971	26	751.88	751.87	752.24	752.87	752.63	752.53	752.17	751.86	751.65	751.53	751.52	751.52
PES. ELEV.	1972	27	751.53	751.54	752.87	752.87	752.80	752.40	752.20	752.07	751.97	751.97	751.97	
PES. ELEV.	1973	28	751.96	751.96	752.01	752.14	752.01	751.85	751.52	751.29	751.07	750.93	750.95	750.96
PES. ELEV.	1974	29	750.96	750.98	752.59	752.87	752.71	752.26	752.19	752.03	751.89	751.89	751.88	
PES. ELEV.	1975	30	751.89	751.91	751.99	752.87	752.87	752.87	752.62	752.36	752.51	752.49	752.49	
PES. ELEV.	1976	31	752.50	752.56	752.87	752.87	752.87	752.87	752.56	752.41	752.41	752.40	752.42	
PES. ELEV.	1977	32	752.42	752.43	752.48	752.49	752.30	752.18	751.86	751.73	751.62	751.49	751.50	751.53
PES. ELEV.	1978	33	751.52	751.52	751.85	751.99	751.85	751.78	751.61	752.01	751.85	751.77	751.77	
PES. ELEV.	1979	34	751.76	751.76	751.97	752.06	752.09	752.14	751.79	751.69	751.53	751.41	751.40	751.40
PES. ELEV.	1980	35	751.39	751.40	751.47	751.52	751.31	751.23	750.82	750.55	750.40	750.28	750.25	750.24
PES. ELEV.	1981	36	750.23	750.20	750.79	750.93	750.70	750.75	750.55	750.29	750.07	749.91	749.89	749.86
PES. ELEV.	1982	37	749.84	749.82	749.88	750.33	750.20	750.08	749.63	749.34	749.10	748.92	748.87	748.84
PES. ELEV.	1983	38	748.81	748.77	748.75	748.75	748.61	748.40	748.01	747.69	747.44	747.27	747.25	747.23
PES. ELEV.	1984	39	747.20	747.20	749.75	749.81	749.70	749.60	749.16	748.88	748.66	748.48	748.46	748.43
PES. ELEV.	1985	40	748.40	748.37	751.68	751.69	751.60	751.76	751.34	751.00	750.75	750.60	750.57	750.56
PES. ELEV.	1986	41	750.53	750.53	750.54	751.32	751.19	751.10	750.77	750.55	750.38	750.25	750.24	750.23
PES. ELEV.	1987	42	750.20	750.18	751.56	751.61	751.47	751.39	751.05	750.87	750.68	750.53	750.54	750.53
PES. ELEV.	1988	43	750.52	750.52	751.63	752.07	751.93	751.97	751.71	751.61	751.57	751.50	751.52	751.54
PES. ELEV.	1989	44	751.54	751.53	752.87	752.87	752.78	752.74	752.26	751.98	751.77	751.70	751.69	751.68
PES. ELEV.	1990	45	751.67	751.67	751.70	751.75	751.69	751.65	751.44	751.30	751.10	750.93	750.95	750.94
PES. ELEV.	1991	46	750.93	750.93	750.93	751.41	751.35	751.28	750.83	750.62	750.49	750.34	750.33	750.33
PES. ELEV.	1992	47	750.31	750.28	751.47	751.50	751.36	751.29	750.95	750.75	750.57	750.46	750.49	750.49
PES. ELEV.	1993	48	750.49	750.47	751.63	752.47	752.74	752.42	752.24	752.08	751.97	751.99	751.99	
PES. ELEV.	1994	49	751.98	751.98	752.07	752.87	752.87	752.77	752.38	752.18	751.94	751.76	751.76	
PES. ELEV.	1995	50	751.75	751.73	752.38	752.76	752.62	752.49	752.15	751.91	751.69	751.59	751.58	751.59
PES. ELEV.	1996	51	751.59	751.58	751.58	751.93	752.87	752.77	752.36	752.23	752.05	751.95	751.95	751.94
PES. ELEV.	1997	52	751.93	751.92	751.93	752.72	752.71	752.39	752.27	752.19	752.02	751.87	751.87	751.85

## 2 UNITS, 1940(GOOD) START, NO SOFTENING, AVERAGE LEACHAGE, RUN 17B.

NAME	YEAR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
T.D.S.	1953	8	698.93	670.37	826.16	760.27	783.92	803.96	673.95	885.81	894.44	897.61	841.86
T.D.S.	1954	9	833.60	785.97	873.56	617.03	668.83	704.37	735.18	886.39	926.06	931.47	878.55
T.D.S.	1955	10	896.75	841.43	702.74	475.12	416.50	765.00	504.6	764.41	831.46	828.97	766.70
T.D.S.	1956	11	596.98	597.05	822.00	754.05	777.13	809.10	636.44	716.05	846.22	882.56	826.30
T.D.S.	1957	12	810.33	767.61	868.22	790.98	818.41	874.02	804.97	924.77	946.66	946.42	895.08
T.D.S.	1958	13	935.26	852.11	685.43	800.75	869.98	907.61	829.96	921.57	966.74	976.08	925.96
T.D.S.	1959	14	950.68	952.81	940.65	848.18	917.75	997.12	1000.49	1047.79	1054.91	1063.47	1021.70
T.D.S.	1960	15	1139.98	1078.71	723.51	741.79	856.82	882.00	799.64	947.92	963.98	973.86	924.06
T.D.S.	1961	16	969.99	885.06	927.96	847.82	868.82	981.43	987.90	1071.72	1048.08	1065.99	1025.41
T.D.S.	1962	17	1196.57	1176.20	944.75	875.29	902.00	902.00	943.58	1017.44	1030.48	1043.28	1001.48
T.D.S.	1963	18	1094.85	973.72	950.79	854.48	884.58	907.75	978.63	1019.68	1044.33	1065.87	1026.17
T.D.S.	1964	19	1140.82	1033.02	1012.72	887.18	922.70	982.99	1154.38	1180.19	1118.26	1146.27	1112.62
T.D.S.	1965	20	1383.74	1404.47	1321.03	902.78	931.79	959.38	1116.97	1123.45	1101.56	1125.15	1087.71
T.D.S.	1966	21	1347.16	1352.17	1020.22	909.99	947.71	1045.72	1220.24	1152.62	1148.14	1173.32	1140.25
T.D.S.	1967	22	1319.08	1330.75	1023.21	896.91	877.51	970.29	1013.16	1079.03	1055.25	1072.07	1030.14
T.D.S.	1968	23	1182.31	1016.76	954.94	860.02	975.09	993.72	1031.67	1001.08	1061.09	1079.88	1040.84
T.D.S.	1969	24	1204.22	1203.98	1173.37	744.73	838.57	944.15	836.61	971.37	985.59	991.10	944.36
T.D.S.	1970	25	994.39	923.82	937.61	752.40	740.19	799.87	785.68	907.93	960.89	968.59	918.85
T.D.S.	1971	26	942.31	901.49	923.15	707.67	830.19	899.37	826.90	953.84	975.24	986.19	938.48
T.D.S.	1972	27	998.23	902.14	699.89	731.45	790.91	845.48	770.49	924.98	946.88	952.48	901.58
T.D.S.	1973	28	935.51	859.99	923.78	832.57	862.47	925.23	963.80	1039.07	1035.58	1047.46	1003.80
T.D.S.	1974	29	1078.05	979.77	920.52	630.40	804.39	829.54	722.19	879.41	920.22	924.59	871.05
T.D.S.	1975	30	904.72	822.86	900.94	464.59	490.09	535.66	601.43	844.06	878.09	877.46	730.50
T.D.S.	1976	31	818.39	755.50	569.56	554.84	687.75	641.36	649.65	872.08	903.90	909.63	859.58
T.D.S.	1977	32	856.30	815.65	897.96	929.44	941.81	929.81	886.82	983.31	990.22	1002.46	956.25
T.D.S.	1978	33	1022.34	924.40	948.98	851.76	984.30	1012.81	980.15	947.59	1002.98	1010.29	963.74
T.D.S.	1979	34	1035.78	965.25	960.10	862.02	887.99	912.55	990.93	1040.64	1079.83	1104.49	1029.49
T.D.S.	1980	35	1103.62	1000.72	1005.19	898.64	1098.25	1196.16	1231.70	1314.93	1257.90	1250.13	1182.24
T.D.S.	1981	36	1350.49	1380.99	1053.20	947.31	1086.34	1001.43	1195.81	1291.20	1385.67	1437.10	1389.34
T.D.S.	1982	37	1501.84	1432.92	1134.65	972.95	1014.79	1365.44	1558.68	1575.53	1559.57	1606.64	1546.71
T.D.S.	1983	38	1649.09	1635.61	1463.92	1215.63	1138.03	1545.94	1848.64	1998.00	1943.49	1671.25	1639.58
T.D.S.	1984	39	2083.33	2023.03	1123.78	990.35	1031.77	1184.81	1488.62	1625.07	1463.84	1398.59	1361.24
T.D.S.	1985	40	1761.43	1791.56	981.82	901.92	980.01	933.23	1084.80	1167.29	1164.12	1210.46	1108.19
T.D.S.	1986	41	1279.62	1182.87	1083.28	903.68	939.15	1022.69	1194.24	1062.54	1170.41	1170.45	1144.93
T.D.S.	1987	42	1369.59	1206.70	1009.22	901.69	953.85	971.59	1163.68	1182.96	1161.28	1162.14	1129.48
T.D.S.	1988	43	1269.54	1138.42	1018.59	892.32	1000.65	1020.24	1091.59	1091.64	1086.03	1104.55	1066.09
T.D.S.	1989	44	1212.05	1072.86	754.10	716.22	864.18	851.34	800.34	886.75	960.88	971.64	923.00
T.D.S.	1990	45	961.29	923.94	981.90	848.43	954.58	903.73	979.65	1004.01	1055.54	1064.94	1020.84
T.D.S.	1991	46	1127.01	1045.44	978.59	876.46	920.05	1021.27	1118.94	1109.11	1092.59	1146.53	1089.24
T.D.S.	1992	47	1241.77	1230.02	986.92	885.07	1033.33	1056.78	1102.51	1224.38	1136.31	1084.71	1181.88
T.D.S.	1993	48	1294.98	1318.93	987.71	802.69	853.24	874.84	896.15	978.34	993.89	1003.32	955.24
T.D.S.	1994	49	1012.67	986.20	958.87	625.60	804.73	879.73	711.87	677.43	910.66	925.07	880.71
T.D.S.	1995	50	844.41	848.89	888.54	801.29	864.98	922.90	808.98	937.41	967.85	977.52	929.54
T.D.S.	1996	51	991.21	994.04	988.26	634.56	807.60	699.81	711.96	900.48	918.89	921.18	867.16
T.D.S.	1997	52	819.00	817.08	921.38	799.60	822.46	852.67	793.53	933.73	940.45	943.24	891.48
T.D.S.	1998	53	867.44	870.74	945.08	433.72	575.02	801.98	544.46	834.28	849.43	849.60	789.05

SUMMARY TABLE FOR T.D.S., RUN #17B

MAY=SEP

AUG-SEP

JL.Y=AUG

JUN-JUN

ANNUAL

SPRING

WINTER

RUNNING

YEAR	WINTER	SPRING	SUMMER	MAY=SEP	JUN-JUN	ANNUAL	JL.Y=AUG	AUG-SEP	JUN-JUN	SPRING	WINTER	YEAR
1953	754.78	771.97	736.31	778.09	762.40	793.48	706.68	718.32	690.05	811.92		
1954	842.68	628.30	731.42	652.28	699.45	677.65	718.38	758.77	904.46	832.50		
1955	719.48	456.71	571.70	496.04	447.38	425.84	547.84	537.78	798.21	869.76		
1956	706.58	765.46	686.50	744.11	718.96	792.10	682.48	648.90	784.62	931.02		
1957	831.76	804.55	845.34	856.12	854.34	843.57	827.23	835.13	936.67	972.19		
1958	896.33	632.93	860.19	883.52	874.96	887.49	847.85	848.94	944.52	986.90		
1959	948.26	880.95	1008.84	981.14	991.05	951.53	999.54	1012.28	1051.55	1001.10		
1960	772.30	772.05	827.77	802.95	843.60	868.81	812.46	819.09	956.01	988.57		
1961	932.70	853.01	998.56	968.43	978.98	915.51	985.92	1004.62	1056.77	981.92		
1962	1053.11	860.10	946.32	961.97	944.09	888.04	933.11	958.27	1024.01	971.09		
1963	1005.66	869.37	973.10	974.66	968.16	895.64	966.77	984.40	1031.61	957.82		
1964	1057.02	904.74	1110.26	1062.96	1072.06	950.56	1097.90	1159.76	1142.93	956.21		
1965	1324.54	917.13	1083.59	1081.65	1054.97	944.96	1077.29	1111.10	1111.45	926.79		
1966	1147.13	928.64	1174.51	1117.43	1130.72	990.75	1178.63	1206.84	1150.25	875.41		
1967	1175.28	896.17	1015.12	937.47	1000.44	917.94	1006.67	1022.04	1066.95	827.83		
1968	1041.14	910.86	1019.00	1013.19	1017.92	984.12	1023.31	1024.45	1028.42	820.80		
1969	1165.93	746.83	865.06	780.44	871.81	881.50	850.79	854.65	978.19	817.15		
1970	955.30	747.51	801.11	786.62	771.08	750.05	789.31	801.48	934.83	827.30		
1971	927.67	714.65	859.03	773.92	868.67	860.80	844.79	848.01	966.47	871.69		
1972	720.36	751.36	800.59	756.31	811.41	813.13	783.10	792.83	935.42	900.85		
1973	913.16	847.36	964.03	946.36	951.99	891.07	951.47	978.66	1037.01	963.07		
1974	993.93	634.44	756.68	671.38	771.90	816.40	740.90	743.38	900.14	1033.99	2	
1975	883.02	468.16	580.47	495.53	546.16	507.08	563.46	633.58	859.57	1044.25	3	
1976	579.21	560.90	649.80	602.33	654.92	645.49	645.38	658.42	885.39	1146.33	4	
1977	867.74	935.67	908.62	927.54	930.10	937.62	894.96	904.59	987.87	1191.25	5	
1978	966.03	907.98	977.07	965.51	981.44	995.53	985.93	971.44	971.49	1208.32	6	
1979	985.55	874.87	982.08	976.73	973.29	899.72	971.20	1000.82	1056.24	1216.95	7	
1980	1039.99	970.36	1238.64	1141.22	1220.97	1140.49	1226.00	1245.42	1288.37	1202.88	8	
1981	1197.12	1008.51	1154.20	1173.61	1160.31	1040.72	1134.53	1212.34	1334.24	1178.02	9	
1982	1315.07	593.64	1528.72	1381.58	1434.60	1153.22	1518.71	1562.26	1569.27	1168.40	10	
1983	1599.92	1173.39	1812.45	1610.75	1661.24	1274.05	1744.04	1872.82	1969.21	1135.38	11	
1984	1628.09	1010.93	1456.13	1377.19	1390.49	1096.83	1422.73	1516.13	1558.13	1059.95	12	
1985	1418.80	938.26	1070.80	1110.99	1066.88	955.83	1058.39	1095.92	1173.89	999.06		
1986	1158.51	921.22	1134.44	1106.94	1104.19	976.68	1152.34	1161.60	1107.19	978.92		
1987	1165.51	927.04	1125.72	1096.95	1100.83	962.47	1115.02	1167.54	1172.87	944.69		
1988	1134.36	941.27	1078.25	1063.10	1067.66	1010.01	1075.79	1091.60	1088.72	918.78		
1989	764.97	726.37	817.64	765.79	832.66	857.71	808.99	811.66	917.93	874.08		
1990	957.07	895.33	964.51	970.07	972.34	927.62	953.15	987.12	1024.31	867.05		
1991	1042.09	897.77	1097.48	1048.29	1064.09	963.64	1095.35	1117.03	1099.97	839.76		
1992	1109.10	947.96	1106.25	1082.48	1103.44	1044.41	1093.29	1116.95	1212.20	778.09		
1993	1131.03	806.30	905.91	864.31	906.89	863.55	890.47	914.66	985.74	739.64		
1994	982.28	628.94	753.09	558.69	771.67	836.88	736.62	734.13	892.10	734.39		
1995	668.74	831.09	847.58	874.86	865.40	898.86	833.70	830.16	954.66	744.72		
1996	844.01	810.90	820.07	852.78	841.81	836.70	808.67	823.98	937.04	766.05		
1997	901.22	436.65	612.22	457.77	620.58	632.79	583.75	582.55	841.91	779.57		
MEAN	1009.31	820.74	956.57	919.85	944.57	890.30	944.25	965.85	1052.79			
PERCENTILES												

2 UNITS, 1948 (GOOD) START, NO SOFTENING, AVERAGE LEACHAGE, RUN 17B.

BORDER

NAME	YEAR	ICYR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
		1953	8	3.04	2.97	2.50	2.18	2.27	2.34	2.07	2.76	2.79	2.88	3.16
BORON	1954	9	3.36	3.09	2.63	1.73	1.72	1.79	1.90	2.51	2.71	2.79	2.93	3.05
BORON	1955	10	3.21	3.16	1.78	1.14	.98	2.12	1.26	2.38	2.47	2.53	2.64	2.70
BORON	1956	11	2.55	2.57	2.35	2.06	2.14	2.27	1.75	2.83	2.86	2.69	2.83	2.91
BORON	1957	12	3.33	3.18	2.50	2.18	2.27	2.51	2.32	2.98	2.83	2.91	3.06	3.21
BORON	1958	13	3.27	3.14	2.57	2.22	2.49	2.66	2.37	2.99	2.91	3.02	3.18	3.45
BORON	1959	14	3.78	3.78	2.76	2.38	2.64	3.06	3.02	3.43	3.29	3.32	3.51	3.75
BORON	1960	15	4.49	4.28	2.09	2.11	2.46	2.57	2.33	2.94	2.94	3.06	3.22	3.39
BORON	1961	16	3.59	3.32	2.74	2.40	2.48	2.98	3.10	4.06	3.22	3.36	3.55	4.04
BORON	1962	17	5.27	5.14	2.80	2.39	2.50	2.61	2.81	3.24	3.18	3.29	3.48	3.72
BORON	1963	18	4.17	3.61	2.82	2.42	2.53	2.62	2.93	3.11	3.21	3.36	3.55	3.80
BORON	1964	19	4.33	3.93	3.03	2.53	2.66	2.90	3.58	4.21	3.46	3.63	3.85	4.66
BORON	1965	20	5.90	6.09	5.28	2.59	2.69	2.80	3.40	3.74	3.41	3.58	3.78	4.08
BORON	1966	21	5.91	5.94	3.06	2.62	2.75	3.16	3.82	3.76	3.58	3.74	3.96	4.31
BORON	1967	22	5.25	5.42	3.08	2.73	2.52	2.92	3.14	3.82	3.26	3.40	3.60	3.85
BORON	1968	23	4.68	3.79	2.85	2.45	2.93	3.00	3.14	3.00	3.28	3.43	3.63	3.89
BORON	1969	24	4.74	4.76	4.45	2.23	2.39	2.92	2.52	3.02	3.14	3.32	3.51	3.51
BORON	1970	25	3.83	3.61	2.79	2.24	2.18	2.33	2.33	3.07	2.95	3.06	3.22	3.39
BORON	1971	26	3.76	3.58	2.74	2.08	2.36	2.66	2.53	3.41	2.99	3.11	3.28	3.47
BORON	1972	27	4.10	3.40	2.06	2.11	2.26	2.43	2.27	2.79	2.90	3.00	3.16	3.32
BORON	1973	28	3.55	3.26	2.74	2.36	2.47	2.73	2.97	3.85	3.30	3.32	3.51	3.98
BORON	1974	29	4.03	3.67	2.74	1.85	2.28	2.38	2.16	2.86	2.82	2.92	3.08	3.22
BORON	1975	30	3.25	3.15	2.68	1.67	1.70	1.73	1.83	2.51	2.71	3.15	3.18	3.17
BORON	1976	31	3.20	2.96	1.79	1.58	1.91	1.57	1.63	2.47	2.78	2.88	3.12	3.07
BORON	1977	32	3.14	2.93	2.55	2.84	2.89	3.65	2.59	3.93	2.99	3.01	3.17	3.33
BORON	1978	33	3.80	3.27	2.71	2.33	2.88	3.37	2.80	2.72	3.06	3.07	3.23	3.56
BORON	1979	34	3.90	3.65	2.77	2.38	2.48	2.57	2.77	3.26	3.74	3.89	3.93	3.82
BORON	1980	35	3.93	3.58	2.93	2.51	3.54	4.40	3.75	4.81	4.53	4.35	4.16	4.35
BORON	1981	36	5.20	5.56	3.10	2.69	3.24	2.89	3.53	4.65	5.21	5.44	5.28	5.86
BORON	1982	37	5.90	5.61	3.39	2.79	2.93	4.57	4.91	5.37	5.55	5.71	5.61	5.96
BORON	1983	38	6.01	5.98	4.97	3.67	3.36	5.25	5.81	7.34	6.90	5.44	5.59	6.32
BORON	1984	39	7.53	7.44	3.40	2.88	3.03	3.66	4.67	5.56	5.09	4.58	4.71	5.58
BORON	1985	40	6.48	6.69	2.95	2.61	2.93	2.73	3.47	4.30	4.46	4.55	4.04	4.14
BORON	1986	41	5.02	4.61	3.34	2.61	2.74	3.09	3.79	3.22	3.86	3.76	4.03	4.81
BORON	1987	42	5.54	4.63	3.05	2.61	2.80	2.87	3.64	4.00	3.84	3.74	3.96	4.31
BORON	1988	43	4.73	4.24	3.08	2.57	3.01	3.11	3.40	3.55	3.40	3.55	3.75	4.04
BORON	1989	44	4.89	4.08	2.32	2.17	2.53	2.46	2.44	3.21	3.41	3.10	3.27	3.45
BORON	1990	45	3.78	3.77	3.25	2.43	2.88	2.64	3.14	3.09	3.73	3.62	3.71	4.53
BORON	1991	46	4.74	4.27	2.95	2.53	2.69	3.15	3.66	3.78	3.44	3.94	3.89	4.14
BORON	1992	47	5.03	5.20	2.99	2.57	3.21	3.36	3.50	4.74	4.96	3.80	3.86	4.53
BORON	1993	48	5.63	5.93	3.01	2.51	2.47	2.56	2.84	3.04	3.16	3.23	3.40	3.70
BORON	1994	49	4.11	4.08	2.91	1.92	2.30	2.67	2.20	2.91	3.14	3.18	3.14	3.52
BORON	1995	50	3.67	3.68	2.65	2.27	2.53	2.96	2.63	3.55	3.20	3.16	3.30	3.69
BORON	1996	51	4.82	4.84	3.64	2.31	2.83	2.19	2.75	2.85	2.94	3.10	3.34	3.41
BORON	1997	52	3.57	3.54	3.14	2.27	2.36	2.49	2.45	2.90	2.92	3.02	3.16	3.14
BORON	1998	53	3.76	3.78	3.30	1.67	1.93	2.48	1.96	2.77	2.77	3.01	3.01	3.14

SUMMARY TABLE FOR BORON, RUN #17B

YEAR WINTER SPRING SUMMER ANNUAL MAY-SEP JUN-JUN JLY-JLY AUG-AUG AUG-SEP RUNNING

1953	2.76	2.72	2.24	2.49	2.31	2.30	2.14	2.22	2.77	2.38
1954	2.99	1.73	1.89	1.80	1.81	1.74	1.84	1.99	2.60	2.43
1955	1.91	1.09	1.51	1.22	1.08	1.01	1.40	1.41	2.43	2.58
1956	2.51	2.10	2.00	2.29	2.11	2.20	1.89	1.92	2.84	2.80
1957	2.86	2.22	2.49	2.62	2.49	2.39	2.38	2.48	2.90	2.94
1958	2.97	2.34	2.53	2.70	2.57	2.57	2.43	2.49	2.95	3.00
1959	3.31	2.50	3.11	3.10	3.04	2.82	3.03	3.12	3.36	3.05
1960	2.35	2.20	2.43	2.41	2.48	2.51	2.36	2.41	2.94	3.01
1961	3.17	2.44	3.21	3.09	3.05	2.69	3.07	3.29	3.53	2.99
1962	3.32	2.45	2.83	3.02	2.82	2.55	2.76	2.90	3.21	2.95
1963	3.45	2.47	2.90	3.01	2.88	2.57	2.88	2.95	3.16	2.91
1964	3.64	2.59	3.49	3.39	3.31	2.77	3.36	3.71	3.76	2.92
1965	5.38	2.64	3.32	3.48	3.22	2.75	3.25	3.46	3.56	2.82
1966	4.14	2.68	3.68	3.59	3.50	2.93	3.67	3.81	3.66	2.68
1967	4.13	2.72	3.19	2.91	3.11	2.69	3.11	3.23	3.54	2.49
1968	3.63	2.66	3.09	3.20	3.09	2.97	3.11	3.10	3.13	2.47
1969	4.44	2.24	2.62	2.38	2.64	2.61	2.57	2.58	3.07	2.45
1970	3.32	2.22	2.40	2.39	2.32	2.21	2.33	2.43	3.01	2.47
1971	3.25	2.10	2.68	2.36	2.66	2.49	2.57	2.68	3.16	2.64
1972	2.19	2.16	2.36	2.28	2.39	2.33	2.30	2.35	2.84	2.73
1973	2.14	2.41	3.03	3.02	2.95	2.59	2.89	3.14	3.53	2.95
1974	3.44	1.86	2.28	2.00	2.33	2.33	2.20	2.26	2.84	3.20
1975	3.02	1.68	1.82	1.74	1.78	1.71	1.77	1.92	2.60	3.40
1976	1.85	1.59	1.61	1.68	1.64	1.60	1.60	1.66	2.60	3.57
1977	2.86	2.86	2.96	2.96	2.96	3.15	2.79	2.83	3.31	3.75
1978	3.16	2.56	2.86	2.93	2.88	3.07	2.90	2.78	2.87	3.80
1979	3.32	2.43	2.80	2.98	2.82	2.52	2.72	2.87	3.45	3.84
1980	3.45	2.89	3.99	3.72	3.98	3.91	3.85	3.93	4.68	3.82
1981	4.13	2.93	3.49	3.81	3.58	3.05	3.33	3.72	4.90	3.73
1982	4.66	2.86	4.94	4.54	4.61	3.58	4.84	5.01	5.44	3.71
1983	5.70	3.50	5.91	5.32	5.39	3.99	5.69	6.06	7.11	3.61
1984	5.57	2.96	4.65	4.47	4.40	3.30	4.45	4.87	5.43	3.35
1985	4.96	2.76	3.45	3.68	3.45	2.83	3.34	3.58	4.36	3.15
1986	4.12	2.68	3.54	3.57	3.43	2.90	3.62	3.65	4.49	3.09
1987	4.16	2.70	3.53	3.53	3.44	2.83	3.44	3.71	3.93	2.98
1988	3.92	2.77	3.37	3.40	3.32	3.06	3.34	3.43	3.47	2.89
1989	2.37	2.19	2.53	2.37	2.59	2.49	2.44	2.54	2.59	2.77
1990	3.53	2.63	2.99	3.20	3.05	2.76	2.96	3.12	3.35	2.75
1991	3.82	2.61	3.58	3.44	3.40	2.89	3.54	3.69	3.59	2.62
1992	3.99	2.84	3.62	3.64	3.65	3.28	3.47	3.68	4.83	3.09
1993	4.21	2.51	2.81	2.76	2.79	2.51	2.76	2.88	3.10	2.24
1994	3.55	1.92	2.34	2.05	2.40	2.46	2.27	2.30	3.01	2.21
1995	3.24	2.40	2.82	2.92	2.81	2.71	2.70	2.78	3.35	2.22
1996	4.16	1.94	2.33	2.13	2.38	2.51	2.27	2.27	2.80	2.25
1997	3.38	2.32	2.54	2.75	2.55	2.42	2.46	2.55	2.91	2.26
1998	3.54	1.68	2.12	1.74	2.12	2.07	2.04	2.07	2.77	2.31
MEAN	3.55	2.40	2.95	2.91	2.90	2.64	2.87	3.00	3.47	

PERCENTILES

2 UNITS. 1948 (GOOD) START. SOFTENING-AVERAGE LEACHAGE-RUN 21B.

NAME	YEAR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
PES. ELEV.	1953	8	752.01	752.02	752.19	752.34	752.34	752.57	752.57	752.21	752.09	752.13	752.16
RES. ELEV.	1954	9	752.16	752.20	752.54	752.87	752.87	752.50	752.45	752.45	752.36	752.38	752.40
RES. ELEV.	1955	10	752.40	752.40	752.87	752.87	752.78	752.48	752.22	752.04	751.94	751.96	751.97
RES. ELEV.	1956	11	751.97	751.97	752.33	752.61	752.52	752.45	752.16	751.97	751.72	751.73	751.75
PES. ELEV.	1957	12	751.74	751.72	751.82	751.92	751.75	751.66	751.40	751.24	751.03	751.06	751.14
RES. ELEV.	1958	13	751.13	751.17	752.14	752.41	752.21	752.05	751.76	751.56	751.38	751.25	751.27
RES. ELEV.	1959	14	751.23	751.19	751.29	751.34	751.18	751.14	750.80	750.55	750.36	750.30	750.32
PES. ELEV.	1960	15	750.32	750.30	752.87	752.87	752.70	752.58	752.16	751.99	751.75	751.61	751.62
PES. ELEV.	1961	16	751.62	751.64	751.97	752.01	751.89	751.70	751.38	751.06	750.82	750.70	750.72
RES. ELEV.	1962	17	750.71	750.69	752.04	752.30	752.17	752.15	751.94	751.74	751.54	751.46	751.45
RES. ELEV.	1963	18	751.45	751.64	752.53	752.66	752.54	752.62	752.19	751.98	751.79	751.61	751.60
PES. ELEV.	1964	19	751.60	751.60	751.62	752.08	751.91	751.80	751.45	751.20	751.07	750.91	750.92
RES. ELEV.	1965	20	750.91	750.89	750.89	751.96	751.97	751.99	751.72	751.58	751.51	751.40	751.43
PES. ELEV.	1966	21	751.42	751.41	752.04	752.08	751.91	751.89	751.55	751.38	751.16	751.08	751.11
PES. ELEV.	1967	22	751.10	751.09	752.21	752.21	752.81	752.62	752.06	751.79	751.61	751.52	751.54
PES. ELEV.	1968	23	751.52	751.51	752.77	752.81	752.63	752.48	752.11	751.97	751.87	751.76	751.77
PES. ELEV.	1969	24	751.76	751.73	751.73	752.08	752.87	752.74	752.21	751.91	751.71	751.67	751.67
PES. ELEV.	1970	25	751.68	751.68	752.11	752.87	752.87	752.45	752.15	751.97	751.87	751.87	751.88
PES. ELEV.	1971	26	751.88	751.87	752.24	752.87	752.63	752.53	752.17	751.86	751.65	751.53	751.52
PES. ELEV.	1972	27	751.53	751.54	752.87	752.87	752.87	752.80	752.40	752.20	752.07	751.97	751.97
PES. ELEV.	1973	28	751.66	751.96	752.01	752.14	752.01	751.85	751.52	751.29	751.07	750.93	750.95
PES. ELEV.	1974	29	750.96	750.98	752.59	752.87	752.87	752.71	752.26	752.19	752.03	751.89	751.88
PES. ELEV.	1975	30	751.89	751.91	751.99	752.97	752.87	752.87	752.76	752.62	752.51	752.49	752.49
PES. ELEV.	1976	31	752.50	752.56	752.87	752.87	752.87	752.87	752.87	752.78	752.56	752.41	752.42
PES. ELEV.	1977	32	752.42	752.43	752.48	752.49	752.30	752.18	751.86	751.73	751.62	751.50	751.53
PES. ELEV.	1978	33	751.52	751.52	751.85	751.99	751.85	751.78	751.61	752.01	751.85	751.77	751.77
PES. ELEV.	1979	34	751.76	751.76	751.97	752.06	752.09	752.14	751.79	751.69	751.53	751.41	751.40
PES. ELEV.	1980	35	751.39	751.40	751.47	751.52	751.31	751.23	750.82	750.55	750.40	750.28	750.25
PES. ELEV.	1981	36	750.23	750.20	750.79	750.83	750.70	750.75	750.55	750.29	750.07	749.91	749.86
PES. ELEV.	1982	37	749.84	749.82	749.88	750.33	750.20	750.08	749.63	749.34	749.10	748.92	748.87
PES. ELEV.	1983	38	748.81	748.77	748.75	748.75	748.61	748.40	748.01	747.69	747.44	747.27	747.25
PES. ELEV.	1984	39	747.20	747.20	749.75	749.81	749.70	749.60	749.16	748.88	748.66	748.48	748.43
PES. ELEV.	1985	40	748.40	748.37	751.68	751.69	751.60	751.76	751.34	751.00	750.75	750.60	750.57
PES. ELEV.	1986	41	750.53	750.54	750.54	751.32	751.19	751.10	750.77	750.55	750.25	750.24	750.23
PES. ELEV.	1987	42	750.20	750.18	751.56	751.61	751.47	751.39	751.05	750.87	750.68	750.53	750.53
PES. ELEV.	1988	43	750.52	750.52	751.63	752.07	751.93	751.97	751.71	751.61	751.57	751.50	751.54
PES. ELEV.	1989	44	751.54	751.53	752.87	752.87	752.87	752.78	752.26	751.98	751.77	751.70	751.69
PES. ELEV.	1990	45	751.67	751.67	751.70	751.75	751.69	751.65	751.44	751.30	751.10	750.93	750.95
PES. ELEV.	1991	46	750.93	750.93	751.41	751.48	751.35	751.28	750.83	750.62	750.49	750.34	750.33
PES. ELEV.	1992	47	750.31	750.28	751.47	751.50	751.36	751.29	750.95	750.75	750.57	750.46	750.49
PES. ELEV.	1993	48	750.49	750.47	751.63	752.87	752.74	752.42	752.24	752.08	751.97	751.99	751.99
PES. ELEV.	1994	49	751.98	751.98	752.07	752.87	752.86	752.77	752.38	752.18	751.94	751.76	751.76
PES. ELEV.	1995	50	751.75	751.73	752.38	752.76	752.62	752.49	752.15	751.91	751.69	751.59	751.59
PES. ELEV.	1996	51	751.59	751.58	751.58	752.87	752.77	752.66	752.36	752.23	752.05	751.95	751.94
PES. ELEV.	1997	52	751.93	751.92	751.93	752.65	752.72	752.71	752.39	752.27	752.19	752.14	752.14
PES. ELEV.	1998	53	752.13	752.11	752.10	752.87	752.78	752.37	752.19	752.02	751.86	751.87	751.85

2 UNITS, 1948 (GOOD) START, SOFTENING, AVERAGE LEACHAGE, RUN 21B.

BORDER

NAME	YEAR	ICYR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
T.D.S.	1953	8	688.40	661.02	621.66	756.83	780.30	800.36	663.83	880.90	889.44	888.82	831.25	836.00
T.D.S.	1954	9	814.86	770.44	664.87	599.48	653.83	691.12	721.68	891.54	920.48	925.33	871.88	886.89
T.D.S.	1955	10	886.66	829.42	690.33	467.10	410.27	763.24	498.41	761.13	828.91	826.20	763.69	696.78
T.D.S.	1956	11	590.33	590.41	819.97	752.53	775.51	807.30	630.82	710.50	842.75	879.18	821.22	823.80
T.D.S.	1957	12	799.72	756.73	863.22	787.13	812.02	864.74	772.29	902.01	923.95	920.93	866.26	879.14
T.D.S.	1958	13	897.57	814.01	866.68	786.71	852.75	867.97	777.73	877.18	934.93	938.00	883.98	899.40
T.D.S.	1959	14	851.46	854.59	908.86	823.26	885.69	953.79	894.41	978.59	989.10	994.54	945.85	976.75
T.D.S.	1960	15	1009.96	954.91	624.64	668.18	826.66	849.33	701.98	904.03	920.35	925.40	871.33	885.30
T.D.S.	1961	16	899.20	821.07	892.35	819.47	838.37	938.59	868.74	966.41	982.12	990.04	941.53	976.99
T.D.S.	1962	17	1014.61	1000.27	896.75	809.53	836.81	861.26	827.34	949.67	968.49	973.72	924.33	950.11
T.D.S.	1963	18	957.21	879.45	903.33	817.97	845.41	866.89	844.01	959.78	976.70	986.58	938.35	966.77
T.D.S.	1964	19	979.83	913.44	947.89	840.61	872.00	923.03	974.27	1037.64	1022.64	1036.45	991.98	1057.40
T.D.S.	1965	20	1122.66	1131.67	1102.27	846.03	872.23	896.70	935.85	1013.68	1011.09	1022.10	975.19	1011.80
T.D.S.	1966	21	1092.60	1096.03	946.99	852.95	885.81	969.06	1001.34	1035.41	1039.59	1051.46	1007.17	1051.80
T.D.S.	1967	22	1112.67	1088.31	944.16	713.99	828.39	907.04	821.42	906.36	975.18	983.89	934.39	962.71
T.D.S.	1968	23	968.17	898.15	899.08	816.43	916.60	932.25	878.53	935.68	981.76	989.27	940.89	970.01
T.D.S.	1969	24	979.91	978.10	994.82	604.36	800.98	889.60	703.78	911.74	920.76	925.89	873.24	888.40
T.D.S.	1970	25	878.53	829.01	890.46	638.54	636.16	734.34	684.20	831.85	915.23	918.23	863.32	874.91
T.D.S.	1971	26	833.62	P11.72	884.67	613.99	803.90	866.86	734.23	889.24	926.26	928.71	874.81	889.52
T.D.S.	1972	27	899.57	823.00	597.75	657.42	744.90	816.77	676.88	887.23	905.95	907.44	851.96	861.53
T.D.S.	1973	28	860.75	796.91	886.29	803.83	829.86	883.98	843.00	934.44	958.40	966.50	915.18	940.73
T.D.S.	1974	29	955.21	870.99	872.81	523.50	776.31	799.11	617.62	823.05	877.19	876.91	819.01	821.74
T.D.S.	1975	30	841.28	759.70	864.07	403.08	436.91	486.99	556.08	827.91	859.01	715.71	686.06	694.20
T.D.S.	1976	31	786.33	727.88	529.27	525.18	668.44	616.79	628.59	865.25	894.41	899.17	846.89	861.93
T.D.S.	1977	32	839.04	603.67	890.91	920.32	932.25	910.85	850.65	945.75	968.01	976.71	926.99	952.26
T.D.S.	1978	33	974.49	886.67	928.02	835.74	960.84	975.56	908.89	925.36	973.15	978.54	928.40	958.47
T.D.S.	1979	34	978.06	911.84	934.52	842.28	866.90	889.76	914.71	986.03	1014.12	1026.06	970.29	1005.50
T.D.S.	1980	35	1022.16	931.70	964.62	866.75	1032.22	1064.36	1064.36	1140.92	1129.77	1128.29	1071.95	1110.89
T.D.S.	1981	36	1183.14	1192.86	992.35	898.89	1021.36	947.99	1054.49	1148.99	1148.99	1239.65	1202.11	1268.03
T.D.S.	1982	37	1277.01	1221.44	1049.77	912.53	949.09	1179.99	1275.28	1307.68	1316.00	1347.41	1301.95	1369.66
T.D.S.	1983	38	1381.60	1345.20	1268.21	1098.09	1030.60	1321.90	1486.83	1607.01	1576.49	1408.69	1368.93	1495.60
T.D.S.	1984	39	1659.58	1599.28	1013.72	905.06	940.19	1063.39	1192.94	1300.80	1246.79	1213.52	1170.58	1287.92
T.D.S.	1985	40	1409.24	1413.07	910.77	843.76	910.15	872.23	860.79	940.14	1006.76	1028.48	980.64	1006.88
T.D.S.	1986	41	1045.70	994.73	997.30	849.21	880.51	952.31	998.04	983.32	1054.23	1058.15	1018.81	1091.52
T.D.S.	1987	42	1143.04	1030.80	938.61	846.82	892.39	907.91	976.60	1035.59	1046.31	1050.79	1007.23	1051.76
T.D.S.	1988	43	1088.22	986.40	945.73	839.70	933.28	949.79	919.86	999.95	1003.96	1012.93	965.69	1000.14
T.D.S.	1989	44	1041.44	942.18	618.11	598.44	827.60	816.37	6H3.60	777.64	887.09	914.68	860.89	873.81
T.D.S.	1990	45	855.87	811.52	920.56	813.64	904.80	864.34	859.34	947.74	963.77	975.78	929.39	964.85
T.D.S.	1991	46	976.11	915.89	922.68	833.20	872.22	929.02	929.73	1003.73	1025.54	1025.88	974.85	1007.80
T.D.S.	1992	47	1053.75	1027.37	921.81	834.25	960.15	976.95	928.68	1008.01	1043.24	1024.09	972.73	1022.71
T.D.S.	1993	48	1071.17	1075.86	922.18	658.62	812.96	832.84	762.62	919.88	929.95	936.15	882.20	897.57
T.D.S.	1994	49	871.94	845.93	907.04	525.80	778.61	845.64	816.92	823.11	857.58	870.34	831.23	790.57
T.D.S.	1995	50	731.78	739.12	856.77	777.90	836.91	883.49	713.84	872.20	914.75	923.68	871.61	876.57
T.D.S.	1996	51	660.16	862.10	912.50	547.33	784.11	609.29	628.53	867.77	883.61	882.34	824.80	822.17
T.D.S.	1997	52	724.80	728.09	877.00	778.43	800.45	828.79	722.63	901.54	906.59	904.30	848.12	853.37
T.D.S.	1998	53	767.02	769.27	894.76	386.66	542.83	787.48	502.32	815.36	630.91	628.96	766.56	758.22

SUMMARY TABLE FOR T.D.S., RUN #21B

YEAR	WINTER	SPRING	SUMMER	ANNUAL	MAY-SEP	JUN-JULY	JLY-AUG	AUG-SEP	RUNNING
1953	747.51	768.43	728.43	770.83	753.55	789.87	698.20	709.29	885.10
1954	829.35	611.30	718.71	636.15	685.57	663.08	705.02	746.63	899.28
1955	707.31	449.26	566.89	488.26	441.43	419.74	542.84	532.43	795.29
1956	702.13	763.89	681.78	740.12	714.95	790.40	677.87	643.29	860.68
1957	824.28	799.44	820.65	837.81	833.71	835.88	802.10	804.95	913.94
1958	865.76	817.41	815.52	848.07	835.23	859.13	803.12	798.33	905.52
1959	884.56	852.67	924.10	914.71	923.59	914.67	911.09	915.40	984.14
1960	671.52	709.87	745.16	719.76	770.81	837.45	724.92	728.47	912.25
1961	877.94	828.91	901.38	899.39	894.84	880.24	890.15	888.22	976.35
1962	949.44	823.03	853.71	883.81	864.02	848.48	835.88	851.67	959.15
1963	919.84	831.54	861.24	884.42	869.41	855.66	847.84	860.26	967.97
1964	952.34	856.14	969.44	954.67	956.76	895.58	957.39	987.49	1028.61
1965	1094.12	858.99	937.95	954.74	935.44	883.91	925.99	949.41	1012.26
1966	1005.52	869.20	1000.26	982.73	984.71	922.37	993.65	1008.09	1037.62
1967	1023.42	718.37	842.78	767.54	853.19	862.66	834.38	832.86	941.32
1968	927.02	860.69	899.16	912.46	911.52	924.19	890.37	892.01	956.67
1969	980.51	608.76	749.99	646.11	771.31	837.01	728.29	731.61	916.06
1970	876.10	637.59	710.43	687.25	681.59	652.39	697.02	703.28	874.21
1971	858.37	624.81	782.93	690.21	805.30	831.76	766.96	760.01	911.09
1972	619.42	686.72	723.43	669.85	742.35	774.16	700.40	707.30	896.15
1973	857.65	816.70	867.33	874.17	871.59	854.52	856.10	861.05	948.58
1974	903.01	529.37	669.05	571.47	702.17	787.19	649.25	645.30	850.55
1975	829.30	407.81	535.09	437.75	497.84	455.58	516.19	592.09	842.08
1976	539.19	531.71	627.25	573.34	632.69	621.39	622.51	637.92	877.45
1977	856.35	926.33	874.99	905.05	903.11	924.79	862.04	862.04	1012.67
1978	933.25	888.80	921.64	926.16	934.09	966.64	920.52	913.15	1046.88
1979	943.79	854.46	920.59	925.77	918.50	877.81	908.43	928.90	945.97
1980	978.18	926.20	1075.25	1031.91	1074.44	1046.08	1064.36	1076.98	1135.73
1981	1083.55	952.80	1036.99	1057.98	1047.49	981.94	1020.92	1070.87	1075.20
1982	1158.80	930.61	1264.75	1183.04	1209.00	1040.24	1255.57	1282.16	1310.94
1983	1338.36	1061.36	1472.33	1347.70	1375.09	1127.74	1451.71	1506.40	1590.90
1984	1351.32	922.43	1187.28	1160.04	1141.98	992.57	1164.84	1214.68	1278.38
1985	1190.42	874.69	871.60	937.89	884.33	890.55	862.79	871.49	966.26
1986	1009.29	864.69	986.16	980.56	974.76	912.76	986.87	994.39	1012.67
1987	1023.96	868.97	971.23	972.10	965.73	899.93	959.20	988.41	1040.58
1988	1006.32	881.99	937.89	951.11	945.32	941.17	926.49	935.15	1002.04
1989	628.90	614.17	714.07	639.81	739.13	821.94	706.11	695.92	823.68
1990	871.27	655.68	880.45	890.99	894.10	885.22	861.09	886.44	954.05
1991	939.02	852.28	946.67	938.32	940.66	909.17	936.19	943.75	1004.73
1992	983.17	887.65	946.65	953.97	955.60	968.09	938.41	940.11	1023.39
1993	992.21	669.63	805.69	741.72	821.10	822.45	781.34	798.04	924.67
1994	886.17	530.52	670.80	563.11	699.22	807.35	650.64	644.64	838.33
1995	794.17	805.51	766.92	812.01	791.84	856.12	750.65	739.94	896.31
1996	883.10	555.76	657.47	599.74	695.42	715.09	626.04	663.59	875.38
1997	795.43	789.32	775.78	805.42	793.40	813.81	749.81	761.50	904.03
1998	826.26	309.91	576.45	411.66	587.07	605.10	545.82	543.46	823.20
MEAN	911.36	764.49	852.05	827.86	853.24	843.78	837.12	849.03	968.63
PERCENTILES									

## 2 UNITS. 1948(GOOD) START, SOFTENING, AVERAGE LEACHAGE, RUN 21B.

BORDER

NAME	YEAR	ICYR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
QORON	1953	8	3.04	2.97	2.50	2.18	2.27	2.34	2.07	2.76	2.79	2.88	3.03	3.16
QORON	1954	9	3.36	3.09	2.63	1.73	1.72	1.79	1.90	2.51	2.71	2.79	2.93	3.05
QORON	1955	10	3.21	3.16	1.78	1.14	.98	2.12	1.26	2.38	2.47	2.53	2.64	2.70
QORON	1956	11	2.55	2.57	2.35	2.06	2.14	2.27	1.75	2.83	2.86	2.69	2.83	2.91
QORON	1957	12	3.33	3.18	2.50	2.18	2.27	2.51	2.32	2.98	2.83	2.91	3.06	3.21
QORON	1958	13	3.27	3.14	2.57	2.22	2.49	2.66	2.37	2.99	2.91	3.02	3.18	3.45
QORON	1959	14	3.78	3.78	2.76	2.38	2.64	3.06	3.02	3.43	3.29	3.32	3.51	3.75
QORON	1960	15	4.49	4.28	2.09	2.11	2.46	2.57	2.33	2.94	2.94	3.06	3.22	3.39
QORON	1961	16	3.59	3.32	2.74	2.40	2.48	2.98	3.10	4.06	3.22	3.36	3.55	4.04
QORON	1962	17	5.27	5.14	2.80	2.39	2.50	2.61	2.81	3.24	3.18	3.29	3.48	3.72
QORON	1963	18	4.17	3.61	2.82	2.42	2.53	2.62	2.93	3.11	3.21	3.36	3.55	3.80
QORON	1964	19	4.33	3.93	3.03	2.53	2.66	2.90	3.58	4.21	3.46	3.63	3.85	4.66
QORON	1965	20	5.90	6.09	5.28	2.59	2.69	2.80	3.40	3.74	3.41	3.58	3.78	4.08
QORON	1966	21	5.91	5.94	3.06	2.62	2.75	3.16	3.82	3.76	3.58	3.74	3.96	4.31
QORON	1967	22	5.25	5.42	3.08	2.73	2.52	2.92	3.14	3.82	3.26	3.40	3.60	3.85
QORON	1968	23	4.68	3.79	2.85	2.45	2.93	3.00	3.14	3.00	3.28	3.43	3.63	3.89
QORON	1969	24	4.74	4.76	4.45	2.23	2.39	2.92	2.52	3.02	3.12	3.14	3.32	3.51
QORON	1970	25	3.83	3.61	2.79	2.24	2.18	2.33	2.33	3.07	2.95	3.06	3.22	3.39
QORON	1971	26	3.76	3.58	2.74	2.08	2.36	2.66	2.53	3.41	2.99	3.11	3.28	3.47
QORON	1972	27	4.10	3.40	2.06	2.11	2.26	2.43	2.27	2.79	2.90	3.00	3.16	3.32
QORON	1973	28	3.55	3.26	2.74	2.36	2.47	2.73	2.97	3.85	3.30	3.32	3.51	3.98
QORON	1974	29	4.03	3.67	2.74	1.85	2.28	2.38	2.16	2.86	2.82	2.92	3.08	3.22
QORON	1975	30	3.28	3.15	2.68	1.67	1.70	1.73	1.83	2.51	2.71	3.15	3.18	3.17
QORON	1976	31	3.20	2.96	1.79	1.58	1.91	1.57	1.63	2.47	2.78	2.88	3.12	3.07
QORON	1931	32	3.14	2.93	2.55	2.84	2.89	3.65	3.65	2.59	3.93	2.99	3.01	3.17
QORON	1932	33	3.80	3.27	2.71	2.33	2.88	3.37	2.80	2.72	3.06	3.07	3.23	3.56
QORON	1933	34	3.90	3.65	2.77	2.38	2.48	2.57	2.77	3.26	3.74	3.89	3.93	3.82
QORON	1934	35	3.93	3.58	2.93	2.51	3.54	4.40	3.75	4.81	4.53	4.35	4.16	4.35
QORON	1935	36	5.20	5.56	3.10	2.69	3.24	2.89	3.53	4.65	5.21	5.44	5.28	5.86
QORON	1936	37	5.90	5.61	3.39	2.79	2.93	4.57	4.91	5.37	5.55	5.71	5.61	5.96
QORON	1937	38	6.01	5.98	4.97	3.67	3.36	5.25	5.81	7.34	6.90	5.44	5.59	6.32
QORON	1938	39	7.53	7.44	3.40	2.88	3.03	3.66	4.67	5.66	5.09	4.58	4.71	5.58
QORON	1939	40	6.48	6.69	2.95	2.61	2.93	2.73	3.47	4.30	4.46	4.55	4.04	4.14
QORON	1940	41	5.02	4.61	3.34	2.61	2.74	3.09	3.79	3.22	3.86	3.76	4.03	4.81
QORON	1941	42	5.54	4.63	3.05	2.61	2.80	2.87	3.64	4.00	3.84	3.74	3.96	4.31
QORON	1942	43	4.73	4.24	3.08	2.57	3.01	3.11	3.40	3.55	3.40	3.55	3.75	4.04
QORON	1943	44	4.89	4.08	2.32	2.17	2.53	2.46	2.44	3.21	3.41	3.10	3.27	3.45
QORON	1944	45	3.78	3.77	3.25	2.43	2.88	2.64	3.14	3.09	3.73	3.62	3.71	4.53
QORON	1945	46	4.74	4.27	2.95	2.53	2.69	3.15	3.66	3.78	3.44	3.94	3.89	4.14
QORON	1946	47	5.03	5.20	2.99	2.57	3.21	3.36	3.50	4.74	4.96	3.80	3.86	4.53
QORON	1947	48	5.63	5.93	3.01	2.51	2.47	2.56	2.84	3.04	3.16	3.23	3.40	3.70
QORON	1948	49	4.11	4.08	2.91	1.92	2.30	2.67	2.20	2.91	3.14	3.18	3.14	3.52
QORON	1949	50	3.67	3.68	2.65	2.27	2.53	2.96	2.63	3.55	3.20	3.16	3.30	3.89
QORON	1950	51	4.82	4.84	3.64	1.92	2.31	2.83	2.19	2.75	2.85	2.94	3.10	3.34
QORON	1951	52	3.57	3.54	3.14	2.27	2.36	2.49	2.45	2.90	2.92	3.02	3.18	3.41
QORON	1952	53	3.76	3.78	3.30	1.67	1.93	2.48	1.96	2.77	2.77	2.87	3.01	3.14

SUMMARY TABLE FOR BORON, RUN #21B

YEAR	WINTER	SPRING	SUMMER	ANNUAL	MAY-SEP	MAY-JUN	JUN-JLY	JLY-AUG	AUG-SEP	RUNNING
1953	2.76	2.22	2.24	2.49	2.31	2.30	2.14	2.22	2.77	2.38
1954	2.99	1.73	1.89	1.80	1.81	1.74	1.84	1.99	2.60	2.43
1955	1.91	1.09	1.51	1.22	1.08	1.01	1.40	1.41	2.43	2.58
1956	2.51	2.10	2.00	2.29	2.11	2.20	1.89	1.92	2.84	2.80
1957	2.86	2.22	2.49	2.62	2.49	2.38	2.38	2.48	2.90	2.94
1958	2.97	2.34	2.53	2.70	2.57	2.43	2.43	2.49	2.95	3.00
1959	3.31	2.50	3.11	3.10	3.04	2.82	3.03	3.12	3.36	3.05
1960	2.36	2.20	2.43	2.41	2.48	2.51	2.36	2.41	2.94	3.01
1961	3.17	2.44	3.21	3.69	3.05	2.69	3.07	3.29	3.53	2.99
1962	3.82	2.45	2.83	3.02	2.82	2.55	2.76	2.90	3.21	2.95
1963	3.45	2.47	2.90	3.01	2.88	2.57	2.88	2.95	3.16	2.91
1964	3.64	2.59	3.49	3.39	3.31	2.77	3.36	3.71	3.76	2.92
1965	5.33	2.64	3.32	3.46	3.22	2.75	3.25	3.46	3.56	2.82
1966	4.14	2.68	3.68	3.58	3.50	2.93	3.67	3.81	3.63	2.68
1967	4.13	2.72	3.19	2.91	3.11	2.69	3.11	3.23	3.54	2.49
1968	3.63	2.66	3.09	3.20	3.09	2.97	3.11	3.10	3.13	2.47
1969	4.44	2.24	2.62	2.38	2.64	2.61	2.57	2.58	3.07	2.45
1970	3.32	2.22	2.40	2.39	2.32	2.21	2.33	2.43	3.01	2.47
1971	3.25	2.10	2.68	2.36	2.66	2.49	2.57	2.68	3.16	2.64
1972	2.19	2.16	2.36	2.28	2.39	2.33	2.30	2.35	2.84	2.73
1973	3.14	2.41	3.03	3.02	2.95	2.59	2.89	3.14	3.53	2.95
1974	3.44	1.86	2.28	2.00	2.33	2.33	2.20	2.26	2.84	3.20
1975	3.02	1.68	1.82	1.74	1.78	1.71	1.77	1.92	2.60	3.40
1976	1.85	1.59	1.61	1.68	1.64	1.60	1.60	1.66	2.60	3.57
1977	2.86	2.86	2.96	2.96	2.96	3.15	2.79	2.83	3.31	3.75
1978	3.16	2.56	2.86	2.93	2.88	3.07	2.90	2.78	2.87	3.80
1979	3.32	2.43	2.80	2.98	2.82	2.52	2.72	2.87	3.45	3.84
1980	3.45	2.88	3.99	3.72	3.98	3.91	3.85	3.93	4.68	3.82
1981	4.13	2.93	3.49	3.81	3.58	3.05	3.33	3.72	4.90	3.73
1982	4.66	2.86	4.94	4.54	4.61	3.58	4.84	5.01	5.44	3.71
1983	5.70	3.50	5.91	5.32	5.39	3.99	5.69	6.06	7.11	3.61
1984	5.57	2.96	4.65	4.47	4.40	3.30	4.45	4.87	5.43	3.35
1985	4.96	2.76	3.45	3.68	3.45	2.83	3.34	3.58	4.36	3.15
1986	4.12	2.68	3.54	3.57	3.43	2.90	3.62	3.65	4.49	3.09
1987	4.16	2.70	3.53	3.53	3.44	2.83	3.44	3.71	3.93	2.98
1988	3.92	2.77	3.37	3.40	3.32	3.06	3.34	3.43	4.47	2.89
1989	3.37	2.19	2.53	2.37	2.59	2.49	2.44	2.54	3.29	2.77
1990	3.53	2.63	2.99	3.20	3.05	2.76	2.96	3.12	3.35	2.75
1991	3.82	2.61	3.58	3.44	3.40	2.89	3.54	3.69	3.59	2.62
1992	3.99	2.84	3.62	3.64	3.65	3.28	3.47	3.68	4.83	3.39
1993	4.21	2.51	2.81	2.76	2.79	2.51	2.76	2.88	3.10	2.24
1994	3.55	1.92	2.34	2.05	2.40	2.46	2.27	2.30	3.01	2.21
1995	3.24	2.40	2.82	2.92	2.81	2.71	2.70	2.78	3.35	2.22
1996	4.16	1.94	2.32	2.13	2.38	2.51	2.27	2.37	2.80	2.25
1997	3.38	2.32	2.54	2.75	2.55	2.42	2.46	2.55	2.91	2.26
1998	3.54	1.68	2.12	1.74	2.12	2.07	2.04	2.07	2.77	2.31
MEAN	3.55	2.40	2.95	2.91	2.90	2.64	2.87	3.00	3.47	
PERCENTILES										
100%	98 %	95 %	95 %	5.58	5.58	3.14				
7.53										

## REPEAT OF 178 WITH DEMREL OBTAINED FROM POPLAR RIV. ••• RUN 24.

NAME	YEAR	ICYR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
PES. ELEV.	1953	8	752.19	752.20	752.37	752.52	752.87	752.83	752.58	752.45	752.33	752.37	752.39	752.58
PES. ELEV.	1954	9	752.40	752.44	752.78	752.87	752.87	752.69	752.64	752.64	752.54	752.57	752.58	752.58
PES. ELEV.	1955	10	752.59	752.59	752.87	752.87	752.78	752.67	752.40	752.40	752.22	752.12	752.14	752.15
PES. ELEV.	1956	11	752.15	752.15	752.51	752.79	752.70	752.62	752.43	752.23	752.05	751.97	751.98	752.01
PES. ELEV.	1957	12	752.00	751.98	752.08	752.17	752.00	751.91	751.70	751.55	751.38	751.40	751.44	751.46
PES. ELEV.	1958	13	751.46	751.49	752.43	752.70	752.50	752.33	752.13	751.92	751.73	751.61	751.63	751.62
PES. ELEV.	1959	14	751.60	751.57	751.65	751.70	751.56	751.52	751.28	751.02	750.83	750.77	750.79	750.80
PES. ELEV.	1960	15	750.79	750.77	752.87	752.87	752.70	752.58	752.35	752.18	751.93	751.79	751.80	751.80
PES. ELEV.	1961	16	751.80	751.82	752.15	752.19	752.07	751.87	751.60	751.32	751.08	750.96	750.96	750.98
PES. ELEV.	1962	17	750.97	750.95	752.25	752.51	752.38	752.36	752.24	752.03	751.82	751.74	751.73	751.73
PES. ELEV.	1963	18	751.73	751.92	752.81	752.87	752.76	752.83	752.59	752.37	752.16	751.98	751.97	751.97
PES. ELEV.	1964	19	751.97	751.97	751.99	751.99	752.45	752.29	752.16	751.85	751.61	751.51	751.38	751.38
PES. ELEV.	1965	20	751.37	751.36	751.36	751.36	752.34	752.35	752.37	752.18	752.04	751.96	751.85	751.88
PES. ELEV.	1966	21	751.87	751.96	752.49	752.53	752.36	752.33	752.08	751.89	751.69	751.62	751.65	751.64
PES. ELEV.	1967	22	751.64	751.63	752.71	752.87	752.81	752.62	752.25	751.98	751.79	751.70	751.71	751.72
PES. ELEV.	1968	23	751.70	751.69	752.87	752.87	752.68	752.53	752.26	752.11	752.02	751.90	751.90	751.92
PES. ELEV.	1969	24	751.90	751.88	751.87	752.87	752.74	752.55	752.40	752.09	751.89	751.85	751.84	751.85
PES. ELEV.	1970	25	751.85	751.86	752.29	752.87	752.87	752.87	752.64	752.33	752.15	752.04	752.05	752.06
PES. ELEV.	1971	26	752.06	752.05	752.42	752.87	752.63	752.53	752.26	751.95	751.74	751.62	751.61	751.61
PES. ELEV.	1972	27	751.61	751.63	752.87	752.87	752.87	752.80	752.59	752.38	752.25	752.15	752.16	752.15
PES. ELEV.	1973	28	752.14	752.14	752.19	752.33	752.19	752.03	751.75	751.52	751.34	751.19	751.21	751.23
PES. ELEV.	1974	29	751.23	751.24	752.81	752.87	752.87	752.71	752.45	752.38	752.21	752.08	752.07	752.06
PES. ELEV.	1975	30	752.08	752.09	752.18	752.87	752.87	752.87	752.87	752.76	752.62	752.51	752.49	752.49
PES. ELEV.	1976	31	752.50	752.56	752.87	752.87	752.87	752.87	752.87	752.78	752.56	752.41	752.40	752.42
PES. ELEV.	1977	32	752.42	752.43	752.48	752.49	752.30	752.18	751.92	751.78	751.68	751.54	751.56	751.59
PES. ELEV.	1978	33	751.58	751.57	751.90	752.05	751.90	751.83	751.76	752.16	752.00	751.91	751.92	751.91
PES. ELEV.	1979	34	751.90	751.90	752.11	752.21	752.24	752.24	752.02	751.93	751.76	751.64	751.62	751.63
PES. ELEV.	1980	35	751.62	751.63	751.70	751.75	751.53	751.47	751.21	750.93	750.78	750.65	750.62	750.61
PES. ELEV.	1981	36	750.61	750.58	751.16	751.20	751.07	751.12	750.99	750.72	750.50	750.33	750.31	750.28
PES. ELEV.	1982	37	750.26	750.24	750.30	750.75	750.61	750.49	750.18	749.94	749.72	749.52	749.48	749.45
PES. ELEV.	1983	38	749.42	749.39	749.36	749.36	749.36	748.98	748.72	748.44	748.22	748.04	748.02	748.01
PES. ELEV.	1984	39	747.98	747.98	750.27	750.32	750.23	750.14	749.99	749.61	749.37	749.18	749.16	749.13
PES. ELEV.	1985	40	749.11	749.08	752.14	752.16	752.06	752.22	751.99	751.68	751.47	751.34	751.31	751.31
PES. ELEV.	1986	41	751.28	751.27	751.29	751.93	751.81	751.73	751.55	751.35	751.17	751.03	751.03	751.01
PES. ELEV.	1987	42	750.99	750.97	752.20	752.25	752.10	752.01	751.80	751.64	751.48	751.35	751.36	751.35
PES. ELEV.	1988	43	751.34	751.35	752.30	752.74	752.60	752.47	752.47	752.35	752.30	752.23	752.25	752.25
PES. ELEV.	1989	44	752.27	752.26	752.87	752.87	752.78	752.74	752.45	752.17	751.95	751.87	751.87	751.86
PES. ELEV.	1990	45	751.85	751.85	751.88	751.93	751.87	751.67	751.53	751.36	751.20	751.22	751.21	
PES. ELEV.	1991	46	751.21	751.20	751.63	751.70	751.57	751.50	751.21	750.99	750.86	750.70	750.69	
PES. ELEV.	1992	47	750.68	750.65	751.77	751.80	751.65	751.59	751.41	751.22	751.03	750.92	750.94	750.95
PES. ELEV.	1993	48	750.94	750.93	752.00	752.07	752.07	752.07	752.51	752.34	752.17	752.06	752.08	
PES. ELEV.	1994	49	752.07	752.07	752.17	752.17	752.86	752.77	752.58	752.36	752.12	751.93	751.94	
PES. ELEV.	1995	50	751.93	751.91	752.56	752.87	752.73	752.60	752.35	752.11	751.88	751.78	751.77	
PES. ELEV.	1996	51	751.78	751.77	751.77	752.87	752.77	752.66	752.42	752.23	752.13	752.13	752.12	
PES. ELEV.	1997	52	752.11	752.10	752.12	752.87	752.86	752.63	752.51	752.37	752.37	752.04	752.04	
PES. ELEV.	1998	53	752.37	752.35	752.34	752.87	752.87	752.56	752.38	752.21				

## REPEAT OF 17A WITH DEMREL OBTAINED FROM POPLAR RIV . . . RUN 24.

BORDER

NAME	YEAR	TCYR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
T.D.S.	1953	8	699.66	670.89	826.29	760.30	783.83	743.40	657.27	885.25	893.67	896.52	840.52	848.45
T.D.S.	1954	9	831.24	784.02	872.70	616.72	668.75	704.30	629.79	886.25	925.72	930.92	877.85	896.09
T.D.S.	1955	10	895.45	839.75	700.82	475.56	416.84	765.09	614.32	764.42	831.32	828.66	766.25	700.68
T.D.S.	1955	11	595.59	595.46	821.60	753.78	776.76	808.64	638.53	713.83	844.49	880.95	824.37	828.30
T.D.S.	1957	12	806.65	763.96	866.57	789.70	816.86	872.09	699.23	921.08	943.15	942.47	890.59	909.95
T.D.S.	1958	13	929.34	846.09	884.09	799.93	868.79	906.15	677.26	917.78	963.92	972.58	921.93	950.06
T.D.S.	1959	14	940.57	942.50	937.39	845.61	914.37	925.53	733.78	1039.66	1046.70	1054.62	1011.83	1058.40
T.D.S.	1960	15	1122.56	1061.83	722.05	747.12	859.01	884.36	634.16	950.96	966.80	976.81	927.16	953.48
T.D.S.	1961	16	973.89	888.50	930.22	849.59	869.96	983.76	719.40	1075.24	1049.65	1067.33	1026.70	1095.00
T.D.S.	1962	17	1198.99	1178.25	947.47	847.79	877.92	904.84	691.74	1021.71	1033.83	1046.64	1004.96	1049.78
T.D.S.	1963	18	1100.41	977.61	954.58	899.32	887.71	911.07	646.20	1023.74	1048.23	1069.77	1030.19	1079.77
T.D.S.	1964	19	1147.44	1037.68	1015.14	890.06	925.57	986.17	769.99	1185.06	1120.86	1148.43	1114.71	1241.31
T.D.S.	1965	20	1387.66	1408.03	1323.46	906.93	936.30	964.21	713.88	1130.93	1107.09	1130.83	1093.65	1157.33
T.D.S.	1966	21	1359.37	1363.79	1026.47	914.83	952.69	1051.81	722.22	1159.70	1153.59	1179.07	1146.26	1223.11
T.D.S.	1967	22	1348.23	1339.95	1032.24	935.02	983.16	647.60	1113.72	1071.11	1089.34	1048.78	1102.52	
T.D.S.	1968	23	1223.27	1039.15	977.20	919.21	988.01	1007.28	706.41	1014.61	1076.84	1097.12	1059.53	1115.91
T.D.S.	1969	24	1245.33	1244.94	1205.50	777.11	847.26	956.76	638.47	984.96	1000.14	1005.58	960.03	994.40
T.D.S.	1970	25	1019.71	944.42	948.35	780.97	767.02	816.77	634.50	927.28	981.02	932.28	959.75	
T.D.S.	1971	26	967.79	922.19	932.40	732.77	837.46	908.36	665.53	970.47	986.71	998.80	952.16	984.99
T.D.S.	1972	27	1021.01	914.61	723.07	748.95	801.79	852.27	633.46	933.78	956.27	962.66	912.56	936.37
T.D.S.	1973	28	951.33	873.09	931.45	838.47	968.72	932.57	740.40	1054.02	1045.55	1057.09	1014.12	1077.64
T.D.S.	1974	29	1092.10	992.15	928.60	653.41	810.51	836.17	625.68	891.56	929.34	934.53	881.81	898.63
T.D.S.	1975	30	917.57	835.55	908.37	482.46	505.67	549.91	597.90	848.79	883.68	883.68	743.51	751.48
T.D.S.	1976	31	827.76	763.57	581.34	563.51	693.39	648.54	643.12	874.07	906.68	912.69	863.29	878.63
T.D.S.	1977	32	861.33	819.15	900.01	932.10	944.59	934.93	668.06	989.04	993.07	1005.25	959.24	932.40
T.D.S.	1978	33	1026.98	927.95	951.08	853.39	986.59	1016.22	663.10	949.95	1005.91	1013.07	966.69	1010.73
T.D.S.	1979	34	1040.22	959.22	962.19	863.66	889.70	914.40	697.26	1043.82	1082.92	1107.99	1031.55	1082.28
T.D.S.	1979	35	1106.17	1002.73	1006.43	899.64	1099.71	1198.74	709.79	1314.25	1256.12	1247.39	1179.36	1232.56
T.D.S.	1979	36	1345.68	1375.05	1053.81	947.81	1086.48	1001.68	782.56	1289.85	1380.34	1429.13	1381.12	1480.43
T.D.S.	1979	37	1490.69	1421.69	1130.50	972.23	1013.47	1360.84	825.24	1561.35	1543.31	1586.29	1526.21	1630.06
T.D.S.	1979	38	1642.95	1608.12	1444.80	1204.11	1126.72	1520.97	901.54	1940.07	1880.37	1621.40	1587.31	1765.01
T.D.S.	1979	39	1999.45	1938.46	1136.00	1000.05	1041.53	1197.27	762.57	1648.87	1475.72	1405.50	1367.14	1554.36
T.D.S.	1979	40	1769.24	1797.58	1066.68	922.07	1003.97	954.62	663.96	1242.38	1240.02	1265.44	1146.03	1186.73
T.D.S.	1940	41	1347.49	1236.65	1107.68	923.15	959.68	1047.07	727.75	1087.03	1203.00	1199.40	1176.49	1318.23
T.D.S.	1941	42	1423.43	1247.42	1036.48	923.00	977.19	995.80	727.16	1234.48	1148.31	1195.38	1165.18	1245.72
T.D.S.	1942	43	1370.45	1180.46	1047.60	915.14	1029.32	1050.19	717.94	1129.09	1119.37	1140.57	1104.82	1170.83
T.D.S.	1943	44	1275.78	1120.91	843.79	794.22	888.40	874.49	639.22	958.60	1009.15	1008.73	963.31	999.24
T.D.S.	1944	45	1029.35	996.32	1021.37	870.82	983.88	928.91	743.07	1036.75	1105.25	1110.31	1066.68	1175.76
T.D.S.	1945	46	1201.17	1108.72	1006.93	898.40	944.11	1052.26	696.21	1153.39	1127.32	1192.12	1131.82	1193.51
T.D.S.	1946	47	1309.99	1302.46	1014.53	906.62	1063.98	1090.07	720.34	1278.64	1290.70	1175.47	1123.59	1236.85
T.D.S.	1947	48	1372.00	1401.50	1014.90	876.32	873.55	896.02	689.83	1007.74	1025.94	1036.90	991.69	1037.99
T.D.S.	1948	49	1082.72	1055.97	984.66	678.86	818.68	897.94	627.71	906.25	938.57	953.60	906.38	917.72
T.D.S.	1949	50	902.43	905.21	905.46	781.33	880.10	944.09	646.86	971.53	992.76	1001.29	954.45	1004.50
T.D.S.	1950	51	1046.55	1049.12	1019.52	676.94	819.16	744.33	629.92	916.46	935.96	949.36	956.11	960.16
T.D.S.	1951	52	863.58	859.11	942.24	810.04	802.16	664.48	677.75	949.36	956.11	960.16	909.79	932.79
T.D.S.	1952	53	908.69	911.77	965.10	460.19	593.20	618.54	844.81	859.57	860.69	860.01	801.01	800.35

SUMMARY TABLE FOR T.D.S., RUN #24

YEAR	WINTER	SPRING	SUMMER	ANNUAL	MAY-SEP	JUN-JUN	JUN-JLY	JLY-AUG	AUG-SEP	RUNNING
1953	755.16	771.94	715.07	767.55	745.48	761.63	683.95	705.01	889.39	757.61
1954	841.10	627.39	687.16	645.29	682.30	677.57	670.41	669.81	904.23	760.90
1955	714.25	457.11	653.37	505.52	460.16	426.17	639.62	633.76	798.14	785.97
1956	705.65	765.15	687.47	743.81	719.42	791.69	683.88	650.31	782.66	827.69
1957	829.20	803.13	785.80	833.74	814.82	841.85	754.96	755.08	933.08	846.88
1958	892.12	831.94	761.42	842.45	803.53	886.17	729.98	727.08	941.22	844.99
1959	941.74	878.00	851.43	924.54	891.78	947.63	806.48	810.05	1043.38	852.05
1960	748.15	776.55	704.51	755.03	740.80	871.08	673.10	675.70	958.94	838.67
1961	935.84	859.77	840.90	920.92	880.14	917.51	800.41	790.36	1059.06	838.91
1962	1055.62	862.69	788.68	896.76	831.64	890.77	745.39	757.37	1027.82	831.09
1963	1009.89	895.78	730.43	862.36	778.38	898.86	690.55	699.19	1035.60	821.94
1964	1061.24	907.62	892.87	991.91	933.04	953.57	841.22	856.58	1146.43	831.35
1965	1327.73	921.45	825.18	970.42	877.30	949.62	776.90	786.56	1117.82	810.70
1966	1155.17	933.56	857.64	993.78	911.41	996.22	800.79	808.90	1156.46	779.15
1967	1483.86	933.69	746.87	925.44	793.91	929.18	698.39	710.40	1092.07	753.49
1968	1029.95	940.44	819.66	948.15	874.10	997.35	772.72	779.14	1042.97	757.74
1969	1199.72	778.59	716.48	791.42	758.01	891.78	680.46	684.85	992.24	754.13
1970	972.75	776.22	705.60	784.94	749.22	775.24	681.12	672.33	950.16	761.91
1971	943.99	737.76	757.46	772.72	801.86	868.84	725.46	716.24	980.06	780.31
1972	741.24	766.65	702.74	749.10	740.22	822.34	670.25	676.89	944.50	797.66
1973	924.84	853.43	837.99	910.33	872.47	897.82	801.84	802.32	1049.02	831.74
1974	1005.35	656.74	688.47	680.84	726.53	822.76	662.36	661.50	910.75	866.33
1975	893.91	485.58	589.36	513.29	561.81	522.16	576.32	618.90	864.69	897.61
1976	590.91	569.43	649.79	509.67	654.80	652.53	645.69	650.88	887.71	924.43
1977	871.06	938.39	760.39	887.82	838.39	941.22	718.56	727.15	991.70	950.31
1978	969.23	909.89	777.45	890.48	837.99	998.26	725.59	739.85	974.09	959.14
1979	988.81	876.54	797.66	897.15	835.80	901.49	751.92	766.19	1059.39	964.87
1980	1041.92	971.52	863.00	997.54	933.20	1142.43	788.21	809.44	1287.18	956.42
1981	1194.82	1008.85	906.52	1068.26	975.42	1040.92	851.64	870.29	1330.54	953.75
1982	1307.36	962.63	1046.15	1172.55	1081.03	1150.60	936.03	981.51	1554.27	945.89
1983	1575.78	1161.98	1153.82	1366.74	1218.34	1258.18	1033.44	1070.66	1908.55	929.39
1984	1587.08	1020.56	987.59	1180.64	1039.29	1107.74	856.85	941.22	1576.98	890.75
1985	1432.59	960.24	774.75	969.78	830.00	978.47	714.61	741.90	1241.46	859.83
1986	1201.50	941.21	861.82	1012.51	913.61	998.93	805.74	816.79	1135.03	856.93
1987	1203.70	949.33	864.40	1005.36	916.70	986.23	795.22	828.74	1217.64	837.81
1988	1173.58	966.74	843.90	988.21	905.31	1039.30	791.51	796.43	1124.03	825.52
1989	851.23	800.68	710.18	834.17	751.33	881.39	679.11	681.03	979.86	801.44
1990	1012.41	920.77	859.07	955.50	906.51	954.73	807.94	833.10	1063.73	800.85
1991	1093.14	920.75	839.79	972.28	896.83	990.69	782.22	785.30	1138.97	778.43
1992	1180.83	876.19	791.01	849.79	831.97	884.28	744.81	761.44	1016.40	715.11
1993	1030.12	681.39	695.46	699.74	730.10	852.66	657.54	665.16	920.52	713.40
1994	907.35	809.00	746.18	841.86	800.94	906.49	711.35	700.39	983.56	720.74
1995	1025.76	681.42	680.07	703.72	722.44	789.62	644.73	671.91	925.83	729.82
1996	896.43	805.28	763.86	831.78	793.83	824.83	725.55	736.75	952.68	732.26
1997	931.76	462.84	670.17	484.50	664.46	648.42	647.77	648.28	852.25	740.89
MEAN	1023.75	835.22	787.86	869.33	825.01	898.77	743.21	753.77	1065.80	

## PERCENTILES

100%	98 %	95 %	50 %
1999.45	1621.40	1423.43	952.69

**REPEAT OF 17B WITH DEMREL OBTAINED FROM POPLAR RIV . . . RUN 24.**

BORDER

NAME	YEAR	ICYR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
BORON	1953	8	3.04	2.97	2.50	2.18	2.27	2.20	1.14	2.76	2.88	3.03	3.15	3.04
BORON	1954	9	3.35	3.08	2.62	1.74	1.72	1.80	.87	2.51	2.79	2.93	2.93	3.04
BORON	1955	10	3.21	3.16	1.78	1.14	.98	2.12	.83	2.38	2.47	2.53	2.64	2.70
BORON	1956	11	2.55	2.56	2.35	2.06	2.14	2.27	1.04	2.82	2.86	2.69	2.82	2.90
BORON	1957	12	3.32	3.17	2.49	2.17	2.27	2.50	1.37	2.97	2.82	2.90	3.05	3.19
BORON	1958	13	3.25	3.12	2.56	2.21	2.48	2.65	1.16	2.97	2.90	3.01	3.16	3.43
BORON	1959	14	3.73	3.74	2.74	2.37	2.63	3.04	1.49	3.40	3.26	3.29	3.47	3.70
BORON	1960	15	4.42	4.21	2.07	2.12	2.46	2.57	.90	2.94	2.94	3.06	3.22	3.40
BORON	1961	16	3.60	3.32	2.75	2.41	2.48	2.99	1.47	4.06	3.22	3.36	3.55	4.04
BORON	1962	17	5.26	5.13	2.81	2.40	2.51	2.61	1.23	3.25	3.19	3.30	3.49	3.72
BORON	1963	18	4.18	3.62	2.83	2.63	2.54	2.63	.94	3.12	3.21	3.37	3.56	3.81
BORON	1964	19	4.34	3.94	3.03	2.54	2.66	2.90	1.62	4.22	3.46	3.63	3.85	4.66
BORON	1965	20	5.89	6.07	5.27	2.60	2.70	2.81	1.31	3.76	3.42	3.59	3.79	4.09
BORON	1966	21	5.93	5.95	3.08	2.63	2.76	3.18	1.33	3.77	3.58	3.75	3.97	4.32
BORON	1967	22	5.26	5.43	3.10	2.84	2.55	2.96	.96	3.92	3.45	3.65	3.92	3.92
BORON	1968	23	4.79	3.85	2.96	2.71	2.97	3.04	1.28	3.03	3.33	3.48	3.68	3.96
BORON	1969	24	4.86	4.87	4.54	2.33	2.42	2.96	.92	3.06	3.16	3.18	3.36	3.57
BORON	1970	25	3.90	3.67	2.82	2.33	2.26	2.38	.91	3.13	2.98	3.09	3.26	3.44
BORON	1971	26	3.83	3.64	2.77	2.16	2.38	2.69	1.14	3.46	3.03	3.15	3.32	3.52
BORON	1972	27	4.16	3.44	2.13	2.17	2.30	2.45	.90	2.81	2.93	3.03	3.19	3.36
BORON	1973	28	3.60	3.30	2.77	2.38	2.49	2.75	1.52	3.90	3.35	3.53	3.63	3.86
BORON	1974	29	4.07	3.70	2.76	1.92	2.30	2.40	.87	2.85	2.95	3.11	3.26	3.26
BORON	1975	30	3.32	3.19	2.70	1.72	1.74	1.76	1.32	2.52	2.73	3.18	3.22	3.20
BORON	1976	31	3.23	2.99	1.82	1.60	1.92	1.59	1.47	2.47	2.79	2.88	3.13	3.08
BORON	1977	32	3.15	2.94	2.55	2.84	2.90	3.66	1.29	3.94	3.00	3.02	3.18	3.34
BORON	1978	33	3.81	3.27	2.72	2.34	2.88	3.38	1.11	2.72	3.07	3.08	3.24	3.57
BORON	1979	34	3.91	3.66	2.77	2.39	2.48	2.58	1.23	3.27	3.74	3.89	3.54	3.82
BORON	1980	35	3.93	3.58	2.93	2.51	3.54	4.40	1.32	4.79	4.52	4.33	4.14	4.33
BORON	1981	36	5.18	5.53	3.10	2.69	3.24	2.89	1.63	4.63	5.18	5.40	5.24	5.80
BORON	1982	37	5.85	5.55	3.37	2.78	2.92	4.54	1.75	5.30	5.48	5.62	5.52	5.85
BORON	1983	38	5.90	5.86	4.89	3.62	3.31	5.14	2.03	7.11	6.65	5.25	5.39	6.07
BORON	1984	39	7.20	7.11	3.43	2.90	3.05	3.69	1.46	5.70	5.10	4.58	4.70	5.56
BORON	1985	40	6.46	6.65	3.02	2.67	3.00	2.80	1.06	4.51	4.61	4.70	4.14	4.25
BORON	1986	41	5.20	4.75	3.41	2.67	2.80	3.15	1.35	3.29	3.95	3.84	4.11	4.93
BORON	1987	42	5.68	4.73	3.13	2.67	2.87	2.94	1.36	4.15	3.94	3.83	4.06	4.43
BORON	1988	43	4.87	4.36	3.17	2.64	3.10	3.20	1.33	3.66	3.50	3.65	3.86	4.18
BORON	1989	44	5.07	4.22	2.59	2.40	2.60	2.54	.92	3.43	3.55	3.21	3.39	3.61
BORON	1990	45	3.99	3.99	3.37	2.50	2.97	2.72	1.56	3.19	3.88	3.75	3.85	4.74
BORON	1991	46	4.96	4.46	3.04	2.59	2.77	3.24	1.26	3.92	3.54	4.08	4.02	4.29
BORON	1992	47	5.23	5.41	3.08	2.63	3.30	3.46	1.34	4.96	5.15	5.91	3.97	4.69
BORON	1993	48	5.85	6.17	3.09	2.75	2.53	2.62	1.23	3.13	3.26	3.33	3.52	3.84
BORON	1994	49	4.32	4.29	2.98	2.08	2.34	2.72	.88	2.99	3.22	3.27	3.22	3.65
BORON	1995	50	3.85	3.85	2.71	2.32	2.58	3.03	1.11	3.66	3.27	3.24	3.37	4.01
BORON	1996	51	4.99	5.01	3.73	2.06	2.34	2.97	.89	2.80	2.90	3.00	3.16	3.42
BORON	1997	52	3.70	3.67	3.20	2.31	2.36	2.52	1.19	2.95	2.07	3.07	3.23	3.18
BORON	1998	53	3.88	3.90	3.36	1.75	2.50	2.80	1.98	2.80	2.90	2.90	2.90	3.05

## SUMMARY TABLE FOR BORON, RUN #24

YEAR	WINTER	SPRING	SUMMER	ANNUAL	MAY-SEP	MAY-JUN	JUN-JULY	JULY-AUG	AUG-SEP	RUNNING
1953	2.76	2.22	1.67	2.23	1.88	2.23	1.47	1.48	2.77	1.91
1954	2.98	1.73	1.46	1.75	1.65	1.74	1.38	1.13	2.60	1.90
1955	1.88	1.09	1.20	1.22	1.04	1.01	1.05	1.03	2.43	1.99
1956	2.51	2.10	1.54	2.11	1.78	2.20	1.36	1.31	2.84	2.12
1957	2.86	2.22	1.96	2.43	2.15	2.37	1.73	1.77	2.89	2.18
1958	2.96	2.34	1.75	2.38	2.01	2.56	1.50	1.54	2.94	2.15
1959	3.29	2.49	2.21	2.79	2.44	2.81	1.93	1.97	3.33	2.17
1960	2.22	2.21	1.36	2.09	1.64	2.52	1.16	1.17	2.94	2.10
1961	3.18	2.44	2.25	2.80	2.45	2.69	1.94	1.99	3.53	2.13
1962	3.82	2.45	1.84	2.60	2.11	2.56	1.58	1.63	3.22	2.09
1963	3.46	2.60	1.45	2.33	1.75	2.58	1.23	1.25	3.16	2.04
1964	3.64	2.60	2.37	3.02	2.59	2.77	2.04	2.16	3.76	2.11
1965	5.37	2.65	1.97	2.89	2.26	2.76	1.69	1.73	3.57	2.01
1966	4.15	2.69	2.08	2.95	2.39	2.94	1.77	1.81	3.67	1.95
1967	4.15	2.83	1.57	2.75	1.87	2.73	1.26	1.36	3.61	1.87
1968	3.41	2.79	1.93	2.79	2.25	3.00	1.67	1.69	3.17	1.92
1969	4.53	2.33	1.41	2.33	1.69	2.64	1.19	1.21	3.11	1.91
1970	3.37	2.31	1.47	2.25	1.94	2.28	1.28	1.19	3.05	1.95
1971	3.30	2.17	1.78	2.27	2.05	2.52	1.53	1.53	3.20	2.02
1972	2.24	2.21	1.37	2.11	1.65	2.36	1.16	1.18	2.87	2.09
1973	3.17	2.43	2.20	2.76	2.41	2.61	1.91	1.99	3.56	2.23
1974	3.48	1.93	1.33	1.95	1.60	2.35	1.13	1.14	2.87	2.35
1975	3.05	1.72	1.57	1.74	1.66	1.75	1.52	1.42	2.61	2.48
1976	1.89	1.62	1.54	1.67	1.57	1.62	1.52	1.50	2.61	2.52
1977	2.87	2.87	2.08	2.71	2.40	3.16	1.74	1.77	3.32	2.60
1978	3.17	2.57	1.79	2.53	2.12	3.08	1.51	1.54	2.87	2.60
1979	3.33	2.44	1.83	2.55	2.09	2.53	1.57	1.63	3.46	2.63
1980	3.45	2.88	2.24	3.05	2.64	3.91	1.82	1.89	4.67	2.59
1981	4.01	2.93	2.36	3.32	2.73	3.05	2.03	2.15	4.88	2.58
1982	4.62	2.85	2.85	3.63	3.08	3.56	2.33	2.50	5.37	2.55
1983	5.59	3.45	3.28	4.35	3.62	3.92	2.70	2.86	6.87	2.49
1984	5.40	2.98	2.57	3.59	2.87	3.32	1.95	2.32	5.45	2.35
1985	4.97	2.82	1.72	2.81	2.05	2.89	1.36	1.52	4.55	2.22
1986	4.24	2.73	2.09	3.02	2.40	2.96	1.79	1.83	3.56	2.23
1987	4.26	2.76	2.13	3.04	2.44	2.90	1.76	1.91	4.05	2.16
1988	4.03	2.85	2.04	2.92	2.38	3.15	1.74	1.77	3.57	2.12
1989	2.63	2.42	1.44	2.52	1.72	2.57	1.19	1.25	3.48	2.04
1990	3.70	2.71	2.24	2.99	2.54	2.84	1.96	2.06	3.46	2.06
1991	3.98	2.68	2.08	2.94	2.40	2.97	1.74	1.78	3.71	1.97
1992	4.12	2.92	2.15	3.09	2.54	3.38	1.77	1.86	5.05	1.83
1993	4.36	2.74	1.87	2.74	2.14	2.58	1.60	1.66	3.19	1.75
1994	3.79	2.09	1.37	2.10	1.64	2.51	1.15	1.16	3.09	1.76
1995	3.36	2.39	1.81	2.57	2.13	2.77	1.52	1.53	3.44	1.79
1996	4.28	2.06	1.37	2.08	1.64	2.59	1.16	1.17	2.85	1.82
1997	3.49	2.34	1.77	2.46	2.04	2.42	1.53	1.57	2.96	1.82
1998	3.63	1.75	1.32	1.77	1.61	2.11	1.13	1.13	2.80	1.86
MEAN	3.59	2.44	1.86	2.59	2.13	2.67	1.59	1.63	3.50	

PERCENTILES

100%	93%	95%	50%	50%
7.20	5.93	5.52	3.11	

## REPEAT OF 17A WITH DEMREL OBTAINED FROM UPSTREAM STORAGE :: RUN 25

NAME	YEAR	ICYR	JAN	FEB	MAR	APP	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
RES. ELEV.	1953	8	752.19	752.20	752.37	752.52	752.52	752.52	752.64	752.64	752.64	752.26	752.29	752.32
RES. ELEV.	1954	9	752.32	752.37	752.70	752.87	752.87	752.87	752.67	752.67	752.67	752.54	752.57	752.58
RES. ELEV.	1955	10	752.59	752.59	752.87	752.87	752.87	752.87	752.40	752.40	752.40	752.12	752.14	752.15
RES. ELEV.	1956	11	752.15	752.15	752.42	752.69	752.61	752.53	752.34	752.34	752.34	751.14	751.97	751.92
RES. ELEV.	1957	12	751.91	751.89	751.93	752.03	751.86	751.77	751.56	751.42	751.22	751.26	751.30	751.33
RES. ELEV.	1958	13	751.33	751.36	751.36	752.20	752.47	752.27	752.11	752.11	751.91	751.53	751.43	751.42
RES. ELEV.	1959	14	751.39	751.39	751.39	751.43	751.29	751.29	751.25	750.97	750.72	750.53	750.47	750.49
RES. ELEV.	1960	15	750.49	750.49	752.87	752.87	752.70	752.59	752.35	752.18	751.93	751.79	751.80	751.80
RES. ELEV.	1961	16	751.80	751.82	752.06	752.06	752.01	751.82	751.55	751.26	751.02	750.90	750.90	750.92
RES. ELEV.	1962	17	750.91	750.89	752.11	752.37	752.23	752.21	752.10	752.10	751.69	751.61	751.60	751.60
RES. ELEV.	1963	18	751.60	751.79	752.49	752.62	752.51	752.58	752.34	752.13	751.93	751.76	751.74	751.75
RES. ELEV.	1964	19	751.74	751.75	751.76	752.17	752.00	751.88	751.58	751.36	751.24	751.08	751.07	751.08
RES. ELEV.	1965	20	751.07	751.06	751.05	751.99	752.01	752.02	751.85	751.71	751.64	751.53	751.55	751.56
RES. ELEV.	1966	21	751.55	751.53	752.07	752.11	751.94	751.92	751.68	751.50	751.30	751.23	751.26	751.25
RES. ELEV.	1967	22	751.25	751.23	752.33	752.33	752.81	752.62	752.25	751.98	751.79	751.70	751.71	751.72
RES. ELEV.	1968	23	751.70	751.69	752.86	752.86	752.68	752.53	752.26	752.11	752.02	751.90	751.90	751.92
RES. ELEV.	1969	24	751.90	751.88	751.87	752.87	752.74	752.40	752.40	752.09	751.89	751.85	751.84	751.85
RES. ELEV.	1970	25	751.85	751.86	752.29	752.87	752.87	752.87	752.64	752.33	752.15	752.04	752.05	752.06
RES. ELEV.	1971	26	752.06	752.05	752.42	752.42	752.63	752.53	752.26	751.95	751.74	751.62	751.61	751.61
RES. ELEV.	1972	27	751.61	751.63	752.87	752.87	752.87	752.80	752.59	752.38	752.25	752.15	752.16	752.15
RES. ELEV.	1973	28	752.14	752.14	752.19	752.27	752.13	751.97	751.70	751.47	751.28	751.13	751.15	751.16
RES. ELEV.	1974	29	751.17	751.18	752.76	752.76	752.87	752.87	752.45	752.38	752.21	752.07	752.07	752.06
RES. ELEV.	1975	30	752.08	752.09	752.18	752.87	752.87	752.87	752.87	752.76	752.62	752.51	752.49	752.49
RES. ELEV.	1976	31	752.50	752.56	752.87	752.87	752.87	752.87	752.87	752.78	752.56	752.41	752.40	752.42
RES. ELEV.	1977	32	752.42	752.43	752.42	752.43	752.24	752.12	751.87	751.73	751.62	751.49	751.51	751.53
RES. ELEV.	1978	33	751.53	751.52	751.85	751.99	751.95	751.85	751.78	751.71	752.01	751.86	751.78	751.77
RES. ELEV.	1979	34	751.76	751.76	751.88	751.97	752.00	752.05	751.79	751.70	751.54	751.42	751.40	751.41
RES. ELEV.	1980	35	751.40	751.41	751.38	751.43	751.20	751.13	750.84	750.56	750.42	750.29	750.26	750.25
RES. ELEV.	1981	36	750.24	750.22	750.73	750.77	750.64	750.69	750.56	750.30	750.09	749.90	749.90	749.88
RES. ELEV.	1982	37	749.85	749.84	749.89	750.22	750.09	749.98	749.65	749.36	749.12	748.94	748.89	748.86
RES. ELEV.	1983	38	748.83	748.80	748.77	748.77	748.55	748.34	748.04	747.72	747.47	747.30	747.28	747.26
RES. ELEV.	1984	39	747.23	747.23	749.63	749.69	749.58	749.48	749.19	748.91	748.69	748.51	748.49	748.46
RES. ELEV.	1985	40	748.43	748.40	751.50	751.52	751.43	751.43	751.36	751.03	750.78	750.62	750.60	750.59
RES. ELEV.	1986	41	750.56	750.56	750.57	751.23	751.10	751.01	750.80	750.58	750.41	750.28	750.27	750.26
RES. ELEV.	1987	42	750.23	750.21	751.49	751.54	751.40	751.32	751.08	750.90	750.71	750.57	750.67	750.68
RES. ELEV.	1988	43	750.68	750.66	751.79	752.87	752.74	752.51	752.34	752.17	752.06	752.08	752.06	752.08
RES. ELEV.	1989	44	751.56	751.56	752.87	752.87	752.78	752.74	752.45	752.17	751.95	751.87	751.87	751.86
RES. ELEV.	1990	45	751.85	751.85	751.88	751.88	751.81	751.78	751.62	751.48	751.31	751.15	751.15	751.15
RES. ELEV.	1991	46	751.14	751.14	751.48	751.56	751.42	751.36	751.04	750.82	750.69	750.54	750.53	750.52
RES. ELEV.	1992	47	750.51	750.48	751.54	751.57	751.42	751.36	751.15	750.95	750.76	750.65	750.67	750.68
RES. ELEV.	1993	48	750.68	750.55	750.56	751.56	752.00	751.87	752.83	752.51	752.34	752.17	752.06	752.08
RES. ELEV.	1994	49	752.07	752.07	752.07	752.17	752.87	752.86	752.58	752.36	752.12	751.93	751.94	751.94
RES. ELEV.	1995	50	751.93	751.91	752.46	752.84	752.70	752.57	752.32	752.08	751.85	751.75	751.74	751.76
RES. ELEV.	1996	51	751.75	751.74	751.74	751.74	752.77	752.66	752.55	752.42	752.23	752.13	752.13	752.12
RES. ELEV.	1997	52	752.11	752.10	752.12	752.74	752.74	752.74	752.57	752.45	752.37	752.31	752.32	752.03
RES. ELEV.	1998	53	752.31	752.28	752.28	752.87	752.87	752.87	752.56	752.38	752.21	752.04	752.04	752.03

## BORDER

## REPEAT OF 178 WITH DEMREL OBTAINED FROM UPSTREAM STORAGE ..RUN 25

J-28

NAME	YEAR	TCYR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
T.D.S.	1953	8	712.47	682.28	931.77	764.50	788.25	808.01	691.20	899.72	903.27	847.88	857.45	
T.D.S.	1954	9	842.84	792.96	877.64	629.19	679.20	713.53	566.57	889.62	929.61	882.49	899.75	
T.D.S.	1955	10	902.47	848.10	709.57	488.66	426.86	767.93	232.99	835.43	833.12	771.09	708.30	
T.D.S.	1956	11	505.27	606.11	826.13	757.17	780.42	812.72	416.76	726.74	852.65	831.52	837.08	
T.D.S.	1957	12	818.76	775.43	872.49	794.27	821.91	878.14	624.68	930.82	951.48	951.51	900.45	
T.D.S.	1958	13	941.52	858.23	892.30	805.94	876.30	914.80	405.45	935.06	975.15	985.21	922.06	
T.D.S.	1959	14	972.74	974.14	949.20	854.41	925.53	1007.28	675.51	1060.56	1065.92	1074.35	1033.38	
T.D.S.	1960	15	1159.21	1095.64	764.91	770.93	868.76	894.92	267.32	965.14	980.89	992.45	944.17	
T.D.S.	1961	16	996.71	909.12	943.72	860.34	881.31	999.60	533.18	1107.58	1068.64	1088.39	1049.67	
T.D.S.	1962	17	1248.29	1225.69	964.14	860.13	891.30	918.98	422.66	1045.44	1054.71	1069.35	1029.83	
T.D.S.	1963	18	1144.16	1006.79	976.59	874.32	905.86	929.94	341.14	1049.87	1076.33	1101.42	1064.90	
T.D.S.	1964	19	1210.57	1084.51	1040.37	909.80	947.18	1011.01	577.26	1239.00	1156.40	1188.35	1158.37	
T.D.S.	1965	20	1481.75	1506.29	1402.20	929.72	960.06	989.14	456.67	1174.87	1142.85	1170.62	1136.93	
T.D.S.	1966	21	1456.88	1461.80	1058.19	939.55	979.75	1085.40	474.76	1208.19	1197.40	1227.12	1198.46	
T.D.S.	1967	22	1436.69	1434.66	1058.77	1004.60	906.51	1007.63	283.76	1180.45	1102.03	1123.38	1085.73	
T.D.S.	1968	23	1305.31	1084.22	992.94	950.00	1014.81	1035.45	437.52	1043.04	1110.19	1133.94	1099.62	
T.D.S.	1969	24	1334.14	1333.79	1275.45	849.01	866.49	984.67	262.52	1015.45	1033.27	1038.88	996.34	
T.D.S.	1970	25	1078.84	992.79	972.46	858.28	836.92	860.80	306.78	978.36	1002.96	1014.80	969.04	
T.D.S.	1971	26	1038.44	979.94	957.20	802.40	856.80	932.27	356.97	1015.03	1017.73	1033.15	989.62	
T.D.S.	1972	27	1083.77	964.02	801.18	804.13	836.07	873.66	281.36	961.89	986.74	996.17	949.04	
T.D.S.	1973	28	1004.72	917.62	957.62	859.69	891.62	959.74	697.60	1112.84	1066.53	1098.33	1058.89	
T.D.S.	1974	29	1153.53	1046.30	951.97	718.38	827.49	854.57	247.60	925.61	955.32	963.31	913.20	
T.D.S.	1975	30	955.82	873.62	930.60	527.71	544.64	585.58	434.92	860.63	897.65	800.23	776.04	
T.D.S.	1976	31	851.21	783.76	610.78	587.31	708.61	667.91	646.75	879.45	914.16	920.93	873.28	
T.D.S.	1977	32	874.91	828.58	906.17	940.09	953.03	950.48	680.53	1007.05	1002.30	1014.59	969.42	
T.D.S.	1978	33	1043.04	940.37	957.86	858.55	994.04	1027.46	674.76	960.06	1019.30	1026.53	981.43	
T.D.S.	1979	34	1063.78	990.92	976.15	874.38	900.94	926.07	542.66	1067.75	1110.10	1139.37	1054.83	
T.D.S.	1980	35	1137.81	1029.20	1026.04	914.82	1129.43	1257.27	654.67	1386.17	1308.30	1296.41	1223.47	
T.D.S.	1981	36	1412.05	1449.54	1080.05	968.67	1115.08	1025.02	614.31	1348.55	1454.60	1510.41	1458.26	
T.D.S.	1982	37	1583.20	1508.86	1164.92	1002.12	1046.44	1452.89	635.03	1691.74	1661.25	1711.60	1644.57	
T.D.S.	1983	38	1782.02	1748.71	1539.30	1260.44	1186.17	1643.45	929.72	2156.66	2087.45	1771.31	1741.35	
T.D.S.	1984	39	2229.55	2178.36	1178.88	1032.96	1077.61	1245.65	489.27	1788.21	1572.77	1488.68	1453.04	
T.D.S.	1985	40	1928.60	1969.15	1028.78	940.21	1026.33	973.46	290.61	1317.70	1301.39	1330.62	1192.33	
T.D.S.	1986	41	1434.06	1306.75	1139.86	945.04	983.75	1076.24	480.62	1120.80	1252.54	1247.26	1230.19	
T.D.S.	1987	42	1520.04	1322.55	1062.95	943.40	1000.63	1020.05	462.19	1293.11	1244.13	1239.81	1213.84	
T.D.S.	1988	43	1392.79	1241.09	1075.00	932.80	1052.53	1074.49	463.29	1149.23	1149.23	1173.77	1141.14	
T.D.S.	1989	44	1337.52	1168.31	872.36	818.54	895.95	881.71	256.61	981.10	1024.35	1020.45	976.09	
T.D.S.	1990	45	1051.03	1019.43	1033.97	879.39	995.21	938.66	692.51	1049.73	1125.29	1128.94	1085.60	
T.D.S.	1991	46	1231.99	1135.12	1023.53	911.23	958.41	1070.81	457.90	1181.79	1150.27	1223.27	1161.22	
T.D.S.	1992	47	1357.95	1353.93	1035.32	922.84	1087.62	1115.95	458.33	1341.88	1348.47	1210.80	1158.60	
T.D.S.	1993	48	1441.53	1476.87	1032.56	917.61	885.24	908.21	376.83	1024.68	1044.47	1056.36	1012.84	
T.D.S.	1994	49	1123.44	1096.45	999.66	720.93	829.65	912.26	233.95	929.05	960.83	976.54	927.12	
T.D.S.	1995	50	949.59	951.15	921.92	825.98	894.53	964.35	367.69	1004.69	1017.28	1024.91	979.36	
T.D.S.	1996	51	1102.20	1104.74	1051.23	728.15	832.78	796.84	251.07	935.42	956.40	962.30	911.92	
T.D.S.	1997	52	918.04	910.59	967.90	824.57	848.46	880.86	424.29	971.51	978.55	984.69	936.57	
T.D.S.	1998	53	969.72	972.45	905.11	492.70	615.37	820.14	248.37	857.82	872.30	874.37	816.45	

SUMMARY TABLE FOR T.D.S., RUN #25

YEAR	WINTER	SPRING	SUMMER	ANNUAL	MAY-SEP	MAY-JUN	JUN-JLY	JLY-AUG	AUG-SEP	RUNNING
1953	764.02	776.24	748.82	746.96	772.72	797.59	720.61	733.07	895.35	693.20
1954	849.09	640.13	665.54	654.61	680.20	687.72	646.69	616.98	907.84	677.16
1955	722.87	468.55	371.99	503.83	425.46	436.60	322.74	302.50	802.84	697.31
1956	713.29	768.66	546.81	693.23	620.41	795.55	522.33	465.26	793.07	732.29
1957	837.85	807.94	748.06	825.84	792.54	847.35	706.40	701.75	942.05	752.37
1958	902.82	838.64	591.79	781.25	664.37	894.21	522.77	515.15	955.42	733.02
1959	952.96	898.12	824.98	927.79	874.36	960.32	768.72	771.51	1063.39	741.05
1960	916.08	796.67	432.35	705.98	534.78	881.22	365.01	358.83	973.08	707.48
1961	954.28	870.83	739.66	901.65	824.08	930.73	676.11	647.72	1082.95	723.63
1962	1085.43	875.54	625.60	839.88	719.08	904.51	547.61	546.53	1050.11	709.16
1963	1039.63	889.92	512.75	759.46	612.25	917.36	439.73	440.63	1062.68	693.24
1964	1102.23	928.29	796.11	983.44	841.75	976.68	720.19	715.31	1189.30	719.71
1965	1409.31	944.72	670.41	925.99	775.28	973.94	590.72	581.86	1157.31	683.67
1966	1211.06	959.43	713.44	961.95	821.18	1026.15	620.33	620.08	1202.47	660.00
1967	1241.38	1000.64	485.22	932.07	594.08	950.56	393.34	404.58	1140.61	644.47
1968	1108.90	972.14	661.24	903.61	764.61	1024.82	569.31	580.41	1073.63	669.60
1969	1272.87	849.41	435.58	823.01	544.71	914.54	357.79	363.31	1023.99	678.03
1970	1013.19	849.64	501.20	792.54	696.28	840.87	448.50	393.57	990.86	700.81
1971	948.92	805.24	566.36	802.91	678.41	890.20	498.95	466.41	1016.63	723.77
1972	821.49	814.83	452.57	744.23	559.81	851.38	380.94	379.78	973.73	748.09
1973	944.32	875.48	824.92	931.04	870.96	922.66	781.41	779.59	1097.32	795.58
1974	1050.63	720.95	418.57	707.31	521.37	840.42	353.37	338.95	940.70	837.85
1975	926.30	530.08	519.82	540.20	538.57	559.90	502.68	470.57	877.49	880.34
1976	625.16	593.31	660.78	630.73	665.93	671.53	656.78	654.57	893.97	888.63
1977	880.26	946.61	774.21	894.41	950.34	952.14	731.62	740.64	1003.91	904.39
1978	980.12	916.02	788.69	901.55	846.93	1007.21	737.17	751.09	985.62	900.70
1979	1007.32	887.52	706.33	867.56	771.51	912.94	639.18	647.10	1084.61	895.56
1980	1067.36	991.92	841.58	1008.98	925.87	1184.58	751.32	775.26	1349.91	871.04
1981	1240.82	1033.12	819.69	1059.55	924.62	1066.69	743.79	741.54	1396.87	868.07
1982	1371.42	1024.04	960.51	1177.07	1034.74	1206.90	804.21	859.35	1679.78	856.27
1983	1702.24	1220.02	1224.43	1469.78	1299.60	1338.66	1081.70	1129.52	2120.11	831.81
1984	1736.05	1055.04	840.66	1169.04	946.25	1149.04	653.32	751.10	1698.76	772.35
1985	1531.97	980.34	513.07	859.53	624.49	999.01	409.60	429.00	1311.30	727.99
1986	1256.80	964.18	724.71	981.81	829.53	1025.30	626.09	639.27	1175.33	733.95
1987	1262.56	971.21	712.12	965.10	814.47	1010.07	603.51	628.56	1270.31	702.67
1988	1224.01	986.91	686.11	944.40	791.49	1063.03	598.63	596.74	1155.49	691.06
1989	883.56	823.85	431.35	828.79	520.37	888.77	362.58	351.46	999.29	660.22
1990	1030.05	930.56	839.06	957.59	890.13	965.22	778.44	802.02	1079.50	684.86
1991	1116.49	934.30	695.27	930.39	803.63	1006.82	605.96	598.96	1164.35	663.27
1992	1190.33	992.74	679.88	945.02	790.15	1101.01	590.92	585.62	1344.75	625.45
1993	1222.72	915.47	607.45	890.58	705.00	896.20	518.50	522.74	1034.10	608.47
1994	1057.93	723.03	403.53	710.12	502.63	865.06	333.94	327.40	943.08	617.23
1995	939.80	858.06	565.09	802.78	681.08	923.32	497.15	472.71	1011.82	635.40
1996	1062.24	731.94	401.55	703.05	510.75	818.59	321.75	351.36	945.50	655.23
1997	936.10	836.39	614.83	789.62	697.36	863.73	541.18	543.17	974.97	657.64
1998	977.08	495.25	394.94	502.73	489.14	667.49	335.59	328.46	865.12	670.21

## PERCENTILES

MEAN 1066.57 862.69 857.05 733.14 921.91 573.46 574.39 1102.20

## REPEAT OF 17B WITH DEMREL OBTAINED FROM UPSTREAM STORAGE .RUN 25

NAME	YEAR	ICYR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC
BORON	1953	8	3.07	3.00	2.51	2.19	2.28	2.35	1.47	2.78	2.80	2.90	3.05	3.18
BORON	1954	9	3.39	3.11	2.64	1.78	1.76	1.83	1.22	2.53	2.73	2.81	2.95	3.07
BORON	1955	10	3.24	3.19	1.81	1.19	1.01	2.13	.50	2.40	2.49	2.54	2.66	2.72
BORON	1956	11	2.59	2.60	2.37	2.07	2.15	2.28	1.02	2.86	2.88	2.71	2.84	2.93
BORON	1957	12	3.36	3.21	2.51	2.19	2.29	2.52	1.60	3.00	2.85	2.93	3.08	3.23
BORON	1958	13	3.29	3.16	2.59	2.23	2.51	2.68	1.01	3.03	2.94	3.05	3.21	3.49
BORON	1959	14	1.65	3.85	2.78	2.40	2.67	3.09	1.76	3.48	3.33	3.36	3.55	3.80
BORON	1960	15	4.55	4.33	2.22	2.21	2.50	2.61	.60	2.99	2.99	3.11	3.28	3.47
BORON	1961	16	3.68	3.40	2.79	2.44	2.52	3.04	1.49	4.18	3.28	3.43	3.63	4.15
BORON	1962	17	5.43	5.30	2.86	2.44	2.55	2.66	1.07	3.33	3.26	3.38	3.57	3.83
BORON	1963	18	4.33	3.72	2.90	2.48	2.60	2.70	.78	3.21	3.31	3.48	3.68	3.96
BORON	1964	19	4.56	4.10	3.12	2.60	2.74	2.99	1.60	4.41	3.58	3.77	4.00	4.89
BORON	1965	20	6.22	6.42	5.55	2.68	2.79	2.90	1.17	3.91	3.55	3.72	3.94	4.28
BORON	1966	21	6.27	6.29	3.19	2.71	2.86	3.29	1.26	3.94	3.74	3.92	4.15	4.54
BORON	1967	22	5.57	5.76	3.19	3.08	2.61	3.04	.70	4.15	3.42	3.57	3.78	4.08
BORON	1968	23	5.08	4.01	2.97	2.80	3.06	3.14	1.13	3.13	3.44	3.61	3.82	4.13
BORON	1969	24	5.17	5.18	4.78	2.58	2.48	3.05	.60	3.17	3.28	3.30	3.49	3.72
BORON	1970	25	4.11	3.84	2.91	2.59	2.50	2.53	.70	3.30	3.09	3.21	3.39	3.60
BORON	1971	26	4.07	3.84	2.85	2.40	2.45	2.77	.95	3.61	3.13	3.27	3.45	3.68
BORON	1972	27	4.38	3.60	2.40	2.35	2.41	2.52	.64	2.91	3.03	3.15	3.32	3.52
BORON	1973	28	3.78	3.45	2.85	2.45	2.57	2.84	1.82	4.10	3.47	3.49	3.69	4.23
BORON	1974	29	4.28	3.89	2.84	2.14	2.36	2.46	.57	3.01	2.94	3.05	3.22	3.39
BORON	1975	30	2.45	3.32	2.78	1.87	1.86	1.88	1.55	2.56	2.77	2.78	3.32	3.30
BORON	1976	31	3.30	3.05	1.92	1.62	1.98	1.66	1.59	2.49	2.82	2.92	3.17	3.12
BORON	1931	32	3.20	2.97	2.58	2.87	2.93	3.72	1.79	4.01	3.03	3.05	3.22	3.39
BORON	1932	33	3.87	3.32	2.74	2.36	2.91	3.42	1.64	2.76	3.12	3.12	3.29	3.64
BORON	1933	34	3.99	3.74	2.82	2.43	2.52	2.62	1.31	3.35	3.84	4.00	3.62	3.92
BORON	1934	35	4.04	3.67	3.00	2.57	3.65	4.60	1.71	5.05	4.70	4.50	4.30	4.51
BORON	1935	36	5.41	5.79	3.19	2.76	3.34	2.97	1.63	4.84	5.44	5.68	5.51	6.13
BORON	1936	37	6.17	5.86	3.49	2.88	3.04	4.86	1.83	5.76	5.89	6.05	5.93	6.33
BORON	1937	38	6.38	6.36	5.22	3.82	3.52	5.58	2.52	7.87	7.38	5.78	5.93	6.74
BORON	1938	39	8.05	7.95	3.59	3.02	3.18	3.86	1.31	6.21	5.45	4.88	5.02	5.99
BORON	1939	40	7.04	7.28	3.10	2.74	3.08	2.87	.78	4.80	4.85	4.95	4.32	4.45
BORON	1940	41	5.53	5.02	3.53	2.75	2.99	3.27	1.28	3.42	4.14	4.02	4.31	5.22
BORON	1941	42	6.04	5.02	3.23	2.75	2.96	3.03	1.21	4.37	4.12	4.00	4.25	4.66
BORON	1942	43	5.14	4.59	3.27	2.71	3.19	3.29	1.20	3.79	3.61	3.78	4.01	4.35
BORON	1943	44	5.31	4.40	2.71	2.51	2.63	2.57	.59	3.53	3.62	3.26	3.44	3.67
BORON	1944	45	4.08	4.09	3.42	2.54	3.02	2.76	1.85	3.25	3.97	3.83	3.93	4.86
BORON	1945	46	5.09	4.57	3.11	2.65	2.82	3.32	1.24	4.03	3.63	4.20	4.14	4.43
BORON	1946	47	5.43	5.62	3.16	2.69	3.40	3.56	1.20	5.21	5.38	4.05	4.11	4.89
BORON	1947	48	6.12	6.46	3.16	2.91	2.58	2.67	1.01	3.20	3.33	3.41	3.60	3.95
BORON	1948	49	4.49	4.45	3.04	2.24	2.39	2.78	.54	3.08	3.31	3.36	3.30	3.78
BORON	1949	50	4.03	4.02	2.77	2.36	2.64	3.11	1.01	3.78	3.37	3.33	3.47	4.16
BORON	1950	51	5.20	5.22	3.85	2.24	2.39	3.16	.58	2.87	2.98	3.06	3.25	3.54
BORON	1951	52	3.91	3.86	3.30	2.36	2.45	2.59	1.08	3.03	3.05	3.16	3.33	3.61
BORON	1952	53	4.11	4.13	3.47	2.05	2.54	2.54	1.20	2.85	2.84	2.95	3.10	3.25

## SUMMARY TABLE FOR BORON, RUN # 25

YEAR	WINTER	SPRING	SUMMER	ANNUAL	MAY-SEP	JUN-JUN	JUN-JUN	JLY-AUG	AUG-SEP	RUNNING
1953	2.78	2.23	1.87	2.34	2.05	2.31	1.69	1.74	2.79	1.94
1954	3.01	1.78	1.63	1.81	1.74	1.78	1.55	1.42	2.62	1.91
1955	1.91	1.13	.95	1.25	1.03	1.04	.77	.74	2.44	2.00
1956	2.53	2.11	1.54	2.12	1.78	2.21	1.36	1.31	2.87	2.12
1957	2.88	2.24	2.10	2.50	2.25	2.39	1.89	1.95	2.92	2.19
1958	2.99	2.36	1.67	2.37	1.96	2.59	1.39	1.43	2.99	2.14
1959	3.36	2.53	2.40	2.89	2.58	2.85	2.14	2.19	3.40	2.16
1960	2.51	2.28	1.15	2.10	1.48	2.55	.92	.92	2.99	2.06
1961	3.24	2.48	2.29	2.85	2.50	2.74	1.97	2.03	3.61	2.12
1962	3.93	2.50	1.76	2.60	2.06	2.61	1.47	1.52	3.29	2.07
1963	3.56	2.64	1.35	2.28	1.68	2.65	1.10	1.12	3.26	2.02
1964	3.79	2.67	2.41	3.11	2.65	2.85	2.06	2.19	3.91	2.11
1965	5.65	2.73	1.92	2.95	2.26	2.84	1.61	1.65	3.71	1.99
1966	4.34	2.78	2.09	3.04	2.43	3.05	1.74	1.79	3.83	1.94
1967	4.35	3.06	1.42	2.88	1.76	2.80	1.05	1.16	3.78	1.87
1968	3.85	2.89	1.87	2.92	2.23	3.10	1.57	1.60	3.27	1.95
1969	4.79	2.57	1.19	2.52	1.54	2.72	.93	.95	3.22	1.97
1970	3.51	2.55	1.38	2.40	2.03	2.51	1.17	1.03	3.19	2.03
1971	3.46	2.40	1.69	2.45	2.01	2.59	1.40	1.39	3.33	2.12
1972	2.53	2.37	1.20	2.23	1.55	2.46	.96	.97	2.97	2.20
1973	3.31	2.51	2.43	2.92	2.60	2.69	2.15	2.27	3.73	2.37
1974	3.63	2.14	1.14	2.12	1.46	2.41	.90	.90	2.97	2.52
1975	3.16	1.87	1.74	1.89	1.81	1.87	1.70	1.63	2.66	2.67
1976	1.98	1.70	1.64	1.76	1.67	1.69	1.62	1.62	2.63	2.68
1977	2.90	2.90	2.44	2.84	2.64	3.20	2.16	2.20	3.37	2.76
1978	3.21	2.59	2.14	2.69	2.38	3.11	1.95	1.94	2.91	2.74
1979	3.19	2.47	1.91	2.62	2.16	2.57	1.64	1.72	3.54	2.74
1980	3.54	2.95	2.59	3.25	2.93	4.06	2.18	2.26	4.89	2.68
1981	4.28	3.01	2.40	3.44	2.80	3.14	2.05	2.18	5.11	2.65
1982	4.45	2.96	3.04	3.86	3.28	3.76	2.46	2.67	5.81	2.61
1983	6.03	3.65	3.79	4.80	4.08	4.21	3.17	3.39	7.61	2.54
1984	5.93	3.10	2.58	3.77	2.94	3.47	1.87	2.30	5.89	2.34
1985	5.31	2.90	1.56	2.82	1.94	2.97	1.14	1.32	4.82	2.19
1986	4.45	2.82	2.09	3.12	2.44	3.06	1.76	1.81	3.72	2.21
1987	4.48	2.85	2.10	3.07	2.44	2.99	1.67	1.84	4.25	2.12
1988	4.22	2.93	1.99	2.98	2.37	3.24	1.67	1.70	3.70	2.07
1989	2.77	2.52	1.22	2.58	1.54	2.60	.93	.98	3.57	2.02
1990	3.78	2.75	2.41	3.10	2.68	2.88	2.17	2.28	3.53	2.07
1991	4.07	2.73	2.10	3.00	2.43	3.04	1.74	1.78	3.81	1.97
1992	4.26	2.99	2.10	3.14	2.53	3.47	1.68	1.78	5.29	1.83
1993	4.52	2.89	1.76	2.84	2.08	2.62	1.45	1.50	3.26	1.76
1994	3.81	2.25	1.13	2.22	1.46	2.55	.87	.88	3.18	1.77
1995	3.49	2.49	1.77	2.64	2.13	2.83	1.46	1.47	3.55	1.82
1996	4.44	2.25	1.17	2.20	1.50	2.70	.91	.92	2.92	1.87
1997	3.62	2.40	1.74	2.51	2.01	2.51	1.47	1.51	3.04	1.87
1998	3.80	1.87	1.57	1.90	1.80	2.18	1.42	1.42	2.84	1.92
MEAN	3.74	2.54	1.98	2.17	2.75	3.58	1.64	1.64	3.63	

## PERCENTILES

100%	98%	95%	50%
8.05	6.35	5.89	3.19





